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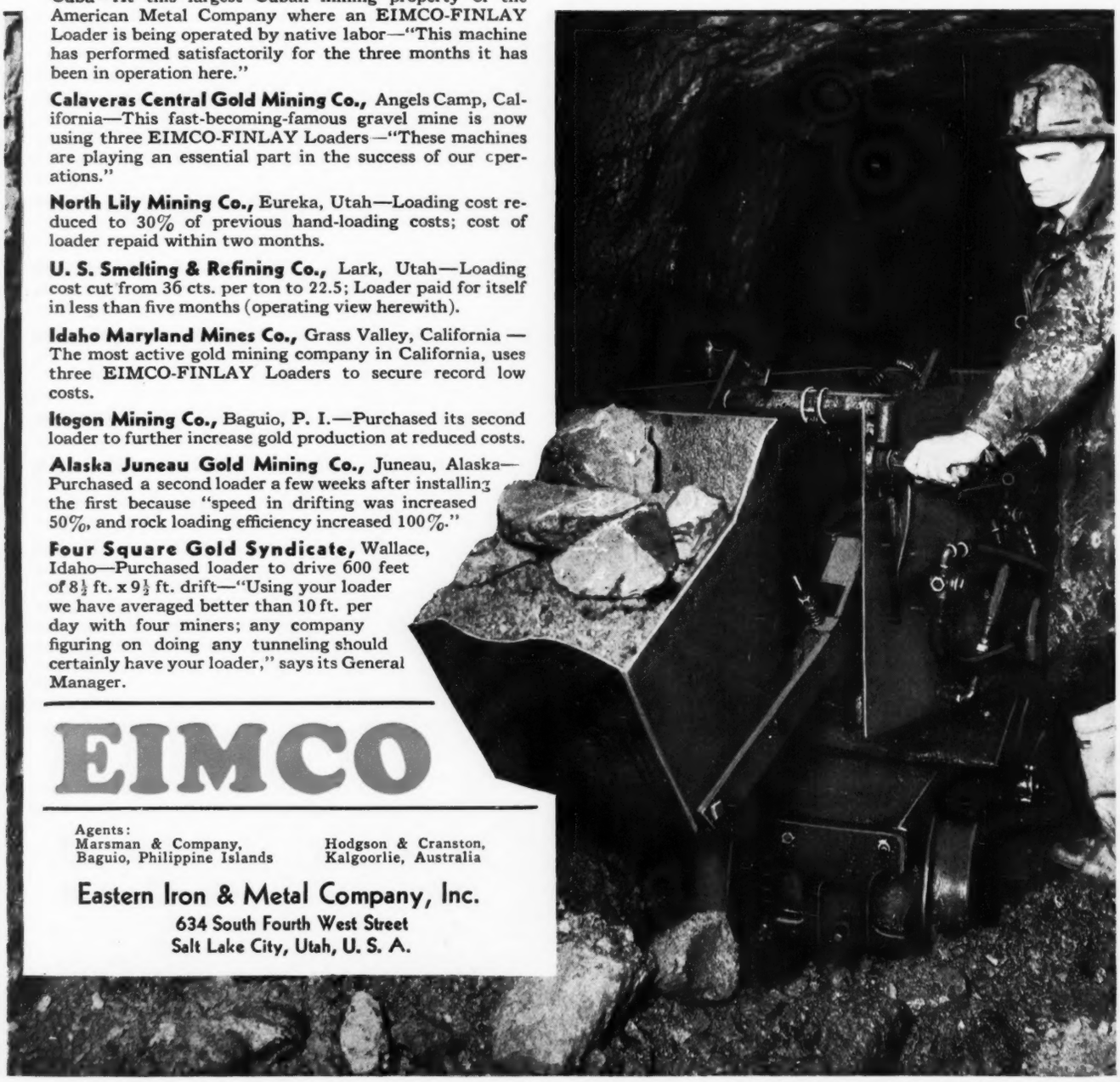

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Engineering and Mining Journal

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March, 1933 Vol. 134, No. 3

Efficient Low-Grade Gold-Ore Treatment

IN COMMENT on an editorial in our esteemed antipodean contemporary, *Chemical Engineering & Mining Review,* of Melbourne, Australia, that complimented us on the Alaska Juneau issue of last September, Mr. B. Du Faur writes in a subsequent issue of that publication to urge more courage in meeting the expense for determining the reserves in potential deposits of low-grade gold ore, in the hope that favorable indications would justify large-scale exploitation and plant equipment. Referring to the low-grade copper enterprises on the American continent, Mr. Du Faur points out that metallurgists, not mining engineers, first recognized the opportunities for successful exploitation, but that persuasive appeal was made for several years before the attention of financiers could be attracted. On one property, Mr. Du Faur adds, about three-quarters of a million dollars was spent to prove grade and extent of ore. "Our American cousins," he avers, "had to take a chance in the initial prospecting stage," but "they won through." He inquires when British or Australian financers will show similar courage with some of the low-grade deposits in Australia.

The Alaska Juneau enterprise, however, indicates the exception rather than the rule governing gold-mining development in the Americas. Mr. Du Faur is correct in stating that the metallurgist had much to do with the development of America's low-grade copper properties, but no similar parallel can be drawn in respect to gold mining. A center of financial activity in which could be found a dozen or more busy copper metallurgists of the highest rank a decade or two ago failed to provide sustenance for a single consultant in the metallurgy of gold. Development of copper recovery techniques well suited to each particular ore resulted; but with gold the situation was entirely different, except in rare instances, of which Alaska Juneau is a conspicuous example. The metallurgist's plea that a low-grade gold ore could be treated at a profit if the problem were handled by an experienced specialist usually fell on deaf ears. Little interest could be stimulated in a gold mine unless high-grade reserves in quantity were blocked out. Rich gold deposits are usually of small extent, and no encouragement was therefore given to the suggestion that a large expenditure be made to develop enough ore to justify the devising of a special treatment technique.

The difference in force of influence between the gold metallurgist and the copper metallurgist has been marked. Whereas the base-metal technician was encouraged to perfect special methods that would be efficient and economical, the directors of most gold-mining enterprises usually decided that a general practitioner would be the best adviser on a highly complicated technical problem; with the result that only in comparatively few instances have sound metallurgical principles been consistently followed—principles that indicated freedom from the influence of prevailing fashion.

Much of the success of low-grade copper ore treatment, particularly in regard to bulk leaching, was the result of the bold adaptation of methods developed by the cyanide metallurgist, who can now turn to the work of his colleagues in the base-metal industries, particularly those responsible for large-scale concentration by flotation, for inspiration and courage to recommend adequate expenditures to determine tonnage and to predict with confidence the profitable treatment of ores of low-grade gold content. The gold metallurgist and ore dresser pioneered in the development of efficient and economical crushing and classification practices, in the mechanical thickening or dewatering of ore pulp and concentrate, in continuous vacuum filtration, in the development of simple and efficient techniques for the extraction of a soluble metal from an insoluble gangue. If his knowledge and experience were utilized to the same extent that the qualifications of the copper metallurgist have been utilized in the last two decades, a new chapter would soon be added to the history of the gold-mining industry.

Shenandoah-Dives Again Makes the Grade

A YEAR AGO, in these columns, attention was drawn to the need to adapt executive direction and operating technique to changed conditions. The history of the Shenandoah-Dives Mining Company was cited as an illustration in proof. A plant was built and put in operation early in 1930 for an expectation of net smelter returns of about $9 per ton milled. Adversity then began to face the company from two directions: the prices of base metals declined, and the ore milled failed to average the grade anticipated by sampling. A new mine had to be operated on less than half the revenue per ton estimated as probable when it was being prepared for production. Despite such adverse factors, however, a profit of 68 cents per ton was made during 1931, by the introduction of general economies, by an increase in the tonnage mined and milled, and by a change in mining methods. A keener foresight was exercised in making provision for mechanical break-down, whereby moderate maintenance cost prevented high repair expense and frequent interruptions to output.

To compare the results of work in 1932, as disclosed in the annual report dated January 25, 1933, with those achieved in 1931 is equally instructive and inspiring, for which tribute is due to Mr. Charles A. Chase, the general manager, and to a staff that has demonstrated its loyalty and efficiency. Perhaps the most significant change of technical interest is the decrease in the assay value of the tailing to 46 cents, from 66 cents in 1931, indicating a substantial improvement in metallurgical

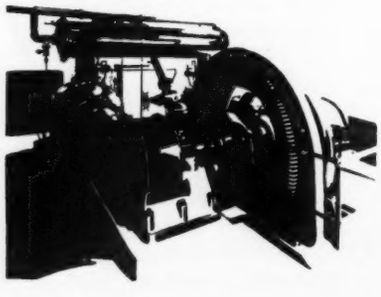
efficiency, although the grade of ore for 1932 was only $4.49 per ton, as compared with $5.98 in 1931. Mr. Chase explains that the result was due to a better control of grinding; moderate losses in the coarser grades of a generally coarser tailing were more than offset by a reduction in the losses formerly incurred in the finer grades.

A year ago *Engineering and Mining Journal* mentioned that Shenandoah-Dives was a gold mine with a byproduct recovery of silver, copper, and lead. The statement is still true. Although the relative proportion of gold to base metals is less than it was, gold still provides the major revenue. Despite the decline in 1932 of over 25 per cent in average value of total metal content of the ore, the company reports an operating margin of 28.6 cents per ton. Successful survival through trying times is mainly attributable to the continuance of a policy that has insured maximum individual and cooperative effort among the members of all departments in the organization.

Mr. Chase pays tribute in his report to the initiative of Mr. Augustus Locke, who originally recommended the studies, later carried out under state and federal direction, that have advanced geological knowledge of the region and led to the prediction that ore will be found far below the existing deepest workings. This work has proved to have been more than justified. With a continuance of competent direction, and a supply of ore adequate for the full capacity of the mill, Shenandoah-Dives faces another year with the hope and courage born of demonstrated character and determination to succeed.

Air Conditioning, Comfort and Output

ATTENTION would be drawn to unhealthful trends if publicity in regard to indoor atmospheric conditions were made obligatory. People become accustomed to change by a gradual alteration in surroundings. They demand the change eventually in larger and larger doses, rarely stopping to analyze cause and effect. For example: An ordinary railroad passenger car in a region of low winter temperature, when the air is naturally dry, is heated to an extent that few would admit, even if confronted with a thermometer. The passengers demand such a temperature. Why? Because they feel cold at a lower one. Again, why? Because the dry air effects an evaporation of moisture from the skin in proportion to the temperature to which it has been raised, and causes an increasing sense of chill. As a result of this reaction, indoor temperatures are boosted to an extent harmful to the delicate mucous membranes of passageways that connect the eyes, ears, nose, and throat. Fuel is wasted. Travelers would realize the need for reform if they were faced with the facts. A large thermometer in a conspicuous place would incite many to think, a few to reason. The result would hasten the day when the treatment of air in public places other than by heating alone would be obligatory.



Business executives will in time realize that an employee using hand or brain, or both, will work at much greater efficiency if the atmosphere is conditioned to prevent excessive dryness or abnormal humidity, both of which are destructive of initiative and sustained performance. All three factors of temperature, moisture content, and air movement must be considered. Scientific conditioning of an indoor atmosphere has been proved practicable and economical for railway passenger equipment, for theaters and hotels. Extension of the innovation to mines, factories, office buildings, apartment houses, and homes will indicate a new conception of architectural or employer obligation, providing significant opportunities for industrial expansion.

Applying Laws of Chance To National Financing

MANY PERSONS—even those who are considered fortunate because they live in a country possessing vast natural resources—are denied the joy of giving without having to count the cost. Opportunity to put promising ideas to the test, to enlarge the mental horizon, to make the acquaintance of the other peoples of the earth, to acquire a knowledge of places beyond one's home town—these come by happy accident, or as a reward for many years of privation and economy. All but a fortunate few are resigned to a future in which luck has little or no scope, in which good fortune has opportunity to come only in rare instances.

Even the humblest sourdough, with no more current assets than a grubstake, and no positive return in prospect for his work, has a healthier outlook on life than has the city laborer in factory or office whose highest expectations can never go beyond "making two ends meet." Throughout the mining industry, in all parts of the world, may be found a wholesome philosophy about the part that hope in the happening of the unexpected plays in improving mental outlook and physical condition. The arguments to be advanced and the recommendations to be made in this editorial are therefore not expected to shock many readers of *Engineering and Mining Journal*.

Never before in the history of the world was the need so apparent for a wider and more active circulation of currency and a rebirth of confidence in the future, both of which can be stimulated by abandoning the pharisaical objections raised to any proposal that frankly and honestly combines investment with a scientific utilization of the laws of chance. Ample money to finance plans to stimulate industrial activity and provide work for jobless men could be secured if the principal invested were guaranteed by the government, and if an additional attraction of regular drawings for prizes were offered to investors. Objection that the issuance of scrip for public-works programs would necessitate so high a rate of interest that the bond market would be demoralized are untenable. Lottery bonds could be issued at an extremely low interest rate, sufficient incentive for purchase and retention being supplied by the condition that each bond, sooner or later, would win a prize.

Unlike gambling in stocks or shares—a vocation or an avocation patronized by many who associate a deterioration of national stamina with official sanction of any type of public "drawing" based on the laws of chance—such a method of raising the spirits of the masses can be operated without fraud or harmful effect; if malfeasance should result, this is merely evidence of official corruption or lack of adequate supervision—either of which characterizes most bank failures in countries where the lottery is prohibited but where banking as a system is not denounced. A lottery-bond issue may be controlled with fairness to all concerned if the methods applied are scientific. Then does it fulfill satisfactorily a common craving among human beings of all nationalities, of all classes of society—the yearning for a personal opportunity now enjoyed only by a privileged few. To deny the practicability of application of the laws of chance to national or local financing is usually evidence of incompetence to cope with a type of fraud that may mar the operation of any respectable business or professional pursuit.

Most people have the gambling instinct, althougn some hesitate to make the admission. Every time an insurance ticket is bought at a railroad station a bet is made with the company. Whenever a pedestrian crosses the street, with the lights or against them, he invites the unexpected. The thread of chance is woven into the whole fabric of human life and endeavor. It follows us from the cradle to the grave. To wait for the happening of the inevitable is to stagnate.

Copper Roof and Wall Construction

AN IMPORTANT DEVELOPMENT that promises an extensive new outlet for copper was brought to a successful issue in 1932, when the problem of producing sound laminate metal by electrolytic deposition on a revolving cathode was solved by technicians and engineers associated with the Raritan Copper Works, a subsidiary of the Anaconda Copper company. These electro sheets, as they are now called, free from pinholes and other imperfections, and presenting the characteristic burnished appearance of rolled copper, are now being produced on a commercial scale, in weights varying from one ounce to eight ounces per square foot, in widths of thirty or fifty inches, and in unlimited lengths.

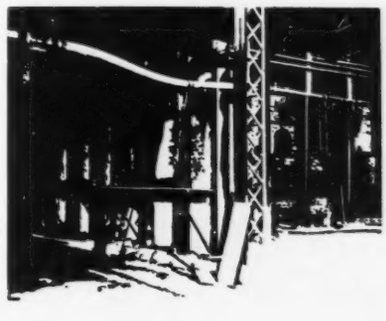
The most promising application of the new form of sheet copper is for use as built-up metal roofing, for the construction of which the American Brass Company has acquired a controlling patent. The metal is used in conjunction with asphalt, which it protects from the destructive action of ultra-violet sun rays; and its use also prevents evaporation of volatile constituents. The asphalt serves as a cushion for the metal, protecting it from damage. In addition to durability and low cost, ease of application, and smooth appearance, the copper provides a covering with an extraordinarily high tensile strength.

From the experience gained by the use of copper as built-up metal roofing was developed another product, consisting of a continuous electro sheet backed and reinforced by a heavy cotton fabric impregnated with asphalt and bonded to the copper by a coating of pitch compound. The combination makes an ideal flashing material, also available for a multitude of construction purposes where impermeability, indestructibility, ease of manipulation, and attractive appearance are needed. In conjunction with a mineral insulating medium, the new fabric-backed copper sheeting may be expected to open up unlimited opportunity for the construction of buildings that will resist loss of heat from within or the passage of heat or cold from without. Much has been written and published about all-steel houses in recent months. The prospects are bright that copper is likely to attract major attention in the future as an element of construction for walls as well as for roofs.

Gold Mining Expansion On the Witwatersrand

DEPARTURE of the government of the Union of South Africa from the gold standard marks an economic paradox: the Witwatersrand district of the Transvaal is by far the leading producer of gold in the world, and local support of gold as a basis for currency and credit would seem the only alternative to intellectual inconsistency. But adversity accepts no law; it recognizes no obligation to observe the niceties of abstract ethics. Export of capital from the Union proved so great a drain that all efforts to remain on the gold standard proved unavailing, and the government was forced to submit to the inevitable.

A year ago *Engineering and Mining Journal* expressed the hope that the problem of treating the lower-grade Witwatersrand ores at a profit and maintaining the gold standard within the Union could and would be solved. To attack the problem from the technical and engineering standpoint was then considered better than to rely on currency debasement, the effect of which, if carried to a logical conclusion, might deprive the Witwatersrand of a market for the product of its mines. Nevertheless, a review of conditions as they today affect the greatest gold field in the world prompts optimistic forecast. Gold production from South Africa in 1932 established a record for the region, and the prospects are bright that the 1933 output will far exceed that of 1932. Many new properties are expected to be equipped for production within the next few years; depreciation in the value of the local currency, in terms of gold, will permit an immediate increase in the available reserves of ore by the inclusion of large amounts that would not pay for extraction and treatment so long as the Union remained on the gold standard. If commodity prices and wages are not raised inordinately, the advantages are all in favor of the mining industry; and a marked stimulation in gold output is likely to result from the government's decision to operate on a sterling basis.



Diamond-Drill Bits and Carbon

W. Rex Storms

IN PROSPECTING for ore by diamond drilling the principal object is to obtain maximum amounts of core and sludge at a minimum cost. Recovery of core and sludge was discussed in a previous article. Low costs involve (1) proper equipment, kept in good condition; (2) good carbon (this includes bort as well as carbon); (3) skilled setters; (4) experienced drillers; (5) close cooperation between setter and driller. The assumption is made that proper equipment will be used and that it will be kept in good condition.

Diamonds occur as (1) white, crystalline diamonds, used for jewelry; (2) minutely crystalline, black, opaque diamonds, or carbonados, called carbons in diamond drilling (including ballas); (3) a less pure white or colored, crystalline form, not worth cutting for jewelry, usually known as bort or bortz. The latter two types are the only ones used in diamond drilling. The black diamond or carbon has been used for diamond drilling since about 1863. It is made up of tiny, impure, diamond crystals; but, unlike the white diamond, it possesses no cleavage. Because of this latter characteristic it does not shatter as easily as the white stone, but wears off when used for drilling. Most carbons are imported from Brazil.

Many companies in the United States sell carbon. However, stones purchased from even the most trustworthy dealers should be selected by an experienced man. They should be carefully examined under a microscope, because a tiny flaw that cannot be seen by the naked eye will cause a weak spot, where breakage may occur. Texture, specific gravity, and, to some extent, color play an important part in the selection of carbon. The stone should not be too porous, or it will wear rapidly when used for drilling. Nor should it be too dense and hard, because then it will break easily. For the same reason, the specific gravity should not be too low nor too high, because a low specific gravity indicates undesirable porosity, whereas a high specific gravity indicates brittleness. Color does not play an important part in the selection of carbon, although an experienced man can differentiate the stones to a great degree by their various colors.

Stones that already are somewhat worn when purchased are usually better than sharp-edged ones, especially for hard-rock drilling. First cost of rounded stones may be higher than for angular ones, but the difference in cost will be saved when actual drilling is started. Angular stones become rounded when used, at the expense of the drilling operation. Rounded stones have no sharp corners to wear off, and can be economically used from the start, but the first run of the angular stones will usually result in a large carbon loss, which in hard rock will continue until the carbons have become rounded and worn. In South America I had the experience of using some large, expensive, angular stones, involving a loss of several hundred dollars during the first few runs. The stones had so many sharp corners that they could not be set without leaving a corner sticking out. In the hard garnet mass of which the rock was composed the first few runs of the drill took off all the sharp corners, and reduced the large stones by many dollars in value and much in weight. Rounded stones generally cost more, but the user saves money in the end. The foregoing discussion deals mostly with large stones of one carat or more in weight. Smaller stones have a lower first cost; therefore a proportionally smaller loss will be had by using stones that are not rounded.

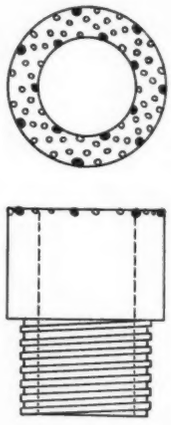


Fig. 1—Many-stone combination bit of bort and carbon. Carbon is shown dark and bort light.

In recent years the trend in diamond drilling has been toward the use of smaller carbon and even scrap carbon, owing to the high price of large diamonds. Average carbon prices during the early part of 1931 were as follows: Large stones of 1½-carat and over, $165 to $175 per carat (200 milligrams); stones of 1¼-carat size, $135 to $150 per carat; those of one-carat size, $115 to $125; three-quarter-carat size, $90 to $100 per carat; one-quarter to one-half-carat size, $35 to $45 per carat; and scrap carbon, $5 to $10 per carat. (Current prices are about 45 per cent lower.)

The high prices of large-sized carbons have caused drillers to seek substitutes. Many varieties of alloy have been used, set in bits in place of black diamonds or bort. Except in soft rock, these metal substitutes are not generally satisfactory. Bort also has been tried and has proved to be a good substitute for high-priced carbon, even for hard-rock drilling. It is used either set in a bit alone or with black diamonds. It possesses cleavage, and therefore is brittle and easily broken. Moses & Parsons, in "Elements of Mineralogy," say: "Bort—Technically any diamond or fragment of diamond not deemed worthy to be cut. Crystallographically, an individual composed of many smaller crystals, sometimes enclosing a simple crystal, oftener not, and having no constant directions of cleavage. Sometimes the little crystals are radially placed and their edges form a rough surfaced sphere."

As with carbon, bort, although relatively low priced, should be selected with care. It is usual practice to sell bort in large lots, the buyer taking them as they come. Nevertheless, much lower drilling costs can be obtained when using bort if the stones are carefully selected one by one. This selection takes time and is a tedious task for the buyer, but it will pay in the end. Broken and irregularly shaped stones should be discarded, and only regular, flawless ones kept. They are usually graded according to size, being so many to the carat, an average size being five or six to the carat. For drilling with a bit of EX size (1$\frac{7}{16}$ in., outside diameter) the smaller sizes should be used, usually seven or eight to the carat; the larger ones can be used in larger bits, such as AX, 1$\frac{27}{32}$ in., outside diameter; BX, 2$\frac{5}{16}$ in.; or NX, 2$\frac{15}{16}$ in. Experience teaches that many small bort in a bit are better than a few large ones. In the spring of 1931 bort cost about $7.50 per carat; the current price is about 10 per cent lower.

Owing to the high price of large-sized carbon and the increasing use of smaller carbon and bort, many new types of bits have been devised, especially in the smaller sizes. A few are shown in Figs. 1, 2, and 3. Fig. 1 shows an EX bit that can be set completely with bort, or it may be a combination of bort and small carbons as shown. Scrap carbons, when set in this type of bit with the more rounded edges upward, serve as well as regular, small-sized carbons, at only a fraction of the cost. Scrap from larger, broken stones is generally used. The scrap carbons help to keep the gage of the bit while the bort do most of the actual cutting. A bit of this type (Fig. 1) has nearly all the available metal space

filled. Practically no metal wear takes place, as was usual in the older type illustrated in Fig. 3. In the type of bit shown in Fig. 1, bort, carbon, or scrap carbon can be set around the bit below the face as reaming stones similar to those in Fig. 2. Or, a reaming shell set with bort or carbon can be used behind the regular bit.

Fig. 2 shows another type of many-stone bit, the one illustrated being a bort bit. This type may also be set as a combination of bort and small carbon. Fig. 1 is shown before water grooves have been cut in the bit. These can be cut by the setter, down the sides of the bit, after setting. In this type the grooves are cut by the setter before he sets the stones. This type of bit is good for soft rocks, such as schists, that disintegrate and produce mud easily when drilled, as the large water grooves keep

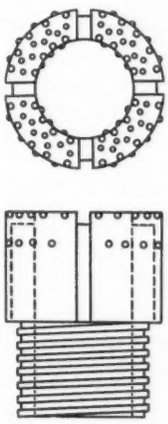


Fig. 2—Another type of many-stone bit. Bit as shown is set with bort.

the bit free from mud even in a very soft formation.

As already mentioned, Fig. 3 shows an eight-stone bit formerly used almost exclusively for drilling. This type of bit loses much metal by wear, as a large free-metal surface occurs between the stones. It is good for soft rock, as is the six-stone bit similar to it, except that it has only three outside and three inside diamonds. This type of bit is usually followed by a reaming shell set with two carbons to keep up the gage of the hole.

Engineering and Mining Journal of Dec. 14, 1931, shows on page 501 two illustrations of the present types of bort bits. One illustration and article show and describe a "round shoulder" bort bit as sold by the Sullivan Machinery Company. This resembles Fig. 1, except that the edges of the bit are beveled with bort set along this rounded edge. Many more stones can thus be set in a standard-sized bit, equalizing the amount of wear and increasing the drilling speed. Incidently, I believe that this type of bit was first developed at the mine of the United Verde Copper Company at Jerome, Ariz., by Mayer G. Hansen, chief mine geologist, and Harry Mader, diamond-drill foreman. Bort bits can now be purchased from the Sullivan company already set and ready to run. Bits set by the Mitchell Diamond-Drill Company, of San Francisco, are also illustrated on the same page of *E.&M.J.* Previous issues of this publication have featured other authoritative articles on the subject of the uses of bort and carbon. The Van Itallie Company, of New York, also sells bort bits. Such bits are usually set in the following manner: The stones are bedded in wax in a special cast, in the shape of a bit; then a molten alloy is poured into this cast. The wax melts and the ready-set bit is left. When a bit is worn out it is returned to the company and another new bit purchased.

Still another new method of bit setting has been tried successfully with bort. In this type all the holes for the track stones in the face of the bit are first bored. The number of holes depends upon the size of bit, as the whole surface should be set with stones, as nearly as possible. These holes, which are a trifle smaller than the bort to be used, are bored slightly deeper than the diameter of the individual stones. Small pieces of lead or copper are then put in the bottom of each hole to insure a snug fit; then the stones are placed in the tops of the holes. A flat piece of soft iron is placed over the top of each stone, and hydraulic pressure applied. The press gently forces the stones into their respective holes, thus setting all track stones at once. Inside and outside stones are now set by hand, all stones are tightened, water grooves are cut, and the bit is finished. A bort bit can be set in this manner in a fraction of the time that it takes to set a bit by hand. Occasionally, a stone is broken in the press, but this seldom happens. Eventually some such method may be developed, for the use of duplex electric setting drills and hydraulic presses, which will enable one setter to set for several drill crews, using bort bits.

Aside from the quality of the carbon itself, low carbon costs depend much upon the diamond setter and on the drill runner, and especially on cooperation between these two men. A setter may have excellent stones, but if they are poorly set the bit will be a failure. Similarly, a good driller cannot obtain footage at low cost if his bits are improperly set. To have an excellent diamond setter on the job, even if he is paid a trifle more than an ordinary one, is usually an economy. A setter who has previously been a good drill runner will have more success than one who has not actually drilled with diamond bits.

With the exception of poor carbon, breakage of carbon by the setter when setting a bit, and a poorly set bit, most of the diamond breakage and high carbon costs can be blamed directly on the driller himself. The major causes of this breakage are:

1. *Running over or grinding out core with the regular carbon bit, instead of putting on a chopping bit and chopping out the lost core*—The driller should carefully mark the rods at the end of a run, or when the bit blocks, before he

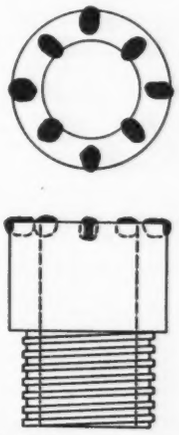


Fig. 3—Eight-stone bit for soft or medium rock.

pulls them. By taking this precaution before the rods reenter the hole, he will know when the rods are on bottom again. Often in a long hole, when the driller finds that he has a small amount of core in the bottom of the hole, he will "take a chance" and try to run over this core with the diamond bit, to get to bottom, instead of pulling the long string of rods and putting down a chopping bit to break up the core.

Carbon Costs, 1928-1931

Drill	Rock	Size	Footage	Angle	Loss, Carats	Cost per Ft.	Bit
Sullivan C Hydraulic.....	Hard, broken diorite.....	A	252	Flat	....	$0.74	(a)Combination
Sullivan C Hydraulic.....	Hard, broken diorite.....	A	264	Flat	....	1.19	(a)Combination
Sullivan C Hydraulic.....	Hard, broken diorite.....	A	59	Flat	....	1.09	(a)Combination
Sullivan C Hydraulic.....	Hard, broken diorite.....	A	38	Flat	....	1.20	(a)Combination
Sullivan C Hydraulic.....	Hard, broken diorite.....	A	413	Flat	....	0.90	(a)Combination
Longyear UG Screw.....	Sulphide, schist, rhyolite...	E	874	Flat	....	0.43	(a)Combination
Longyear UG Screw.....	Sulphide, schist, rhyolite...	E	827	Flat	....	0.38	(a)Combination
Longyear UG Screw.....	Sulphide, schist, rhyolite...	E	815	Flat	....	0.35	(a)Combination
Sullivan C Hydraulic.....	Greenstone.....	A	292	Flat	0.57	1.48	Carbon
Longyear UG Screw.....	Hard sulphide.....	E	298	Flat	1.57	0.80	Carbon
Longyear UG Screw.....	Hard sulphide.....	E	582	Flat	16.31	0.71	Carbon
Sullivan C Hydraulic.....	Hard rhyolite.....	A	434	minus 90 deg.	23.81	1.10	(a)Combination
Longyear UG Screw.....	Schist.....	E	156	Flat	2.23	0.24	(a)Combination
Longyear UG Hydraulic.....	Greenstone and rhyolite...	A	410	Minus 40 deg.	7.38	0.28	(a)Combination
Longyear UG Screws.....	Hard sulphides.....	E	65	Flat	2.53	0.89	(a)Combination
Longyear Junior.....	Rhyolite.....	E	39	Flat	2.96	0.73	(a)Combination

(a) Combination bit of bort and carbon. Cost per foot is for carbon (and bort) only; it does not include cost of setting bits.

2. *Running into broken, fractured rock after having had good drilling*—In this event, even if no stone is broken in drilling, one is often scraped from the bit when the driller attempts to pull the rods. Encountering bad ground cannot be blamed on the driller, because he usually has orders to drill the hole to a certain depth if possible, but good judgment on his part while drilling bad ground will result in less carbon loss.

3. *Rod vibration or chattering*—This condition may cause excessive carbon breakage, especially in a deep hole when the driller is forcing the machine.

4. *Dry blocking*—Unless a driller is expert at dry blocking he should not attempt it, but should use a core lifter. Many good bits have been "burned in" and lost by inexperienced drillers when trying to dry-block a core. Dry blocking is done by shutting off the water from the drill rods and drilling for a short time with the hole dry. This practice muds up the bit and plugs it so tight that the core cannot drop out. An experienced driller knows just how long he can drill dry for a good block and not "burn the bit in." A "burned-in" bit will become so hot from drilling without water that it will actually fuse into the rock and stick.

5. *Dropping rods in a down hole*—This will usually smash most of the stones in the bit.

6. *Removing rods through bad casing*—This example is similar to pulling rods through broken rocks, as the outside stones in the bit may catch on the casing joints and pull out.

Many other causes of diamond breakage could be named, but those mentioned, especially the first two, account for the largest amount of loss.

Legitimate carbon losses vary with the kind of rock drilled. Table I gives costs for carbon when drilling in various kinds of rock. These costs are given with the explanation that good carbon and equipment was used, and that the setters and drillers were of average grade, or a trifle above average. Carbon prices were about the same per carat as for 1931.

Consumption of Flotation Reagents

MORE than 83 per cent of the total amount of non-ferrous ore mined in the United States during 1931, or 45,591,896 tons, was treated in concentrations plants; 2,094,482 tons, or 3.8 per cent, was shipped directly to smelters; 3,428,960 tons, or 6.26 per cent, was treated in gold and silver mills; and the remaining 3,649,504 tons, or 6.66 per cent, was treated in miscellaneous plants, including copper leaching plants and magnetic-separation plants. A detailed classification of the ores treated is given in Table I.

Table I—Ores Treated

Class of Ore Treated	Ore Treated, in 1,000 Tons: Straight Gravity	Combined Gravity and Flotation	Straight Flotation
Copper	1	5,053	25,076
Copper-lead	...	...	195
Lead	10	5,652	236
Lead-zinc	102	4,120	1,543
Zinc	69	2,775	357
Gold-silver	4	1	397
Totals	186	17,601	27,804

Table II gives the itemized reagent consumption for all ores treated. The total quantity of reagents used in 160 plants treating 35,955,669 tons of ore was 143,068,309 lb., or 3.979 lb. per ton of ore, compared with 4.036 lb. per ton in 1930. Of this amount, frothing reagents accounted for 5,507,768 lb., or 0.153 lb. per ton (0.152 lb. in 1930); collectors totaled 4,295,744 lb., or 0.119 per ton (0.130); acids, 11,142,955 lb. or 21.342 lb. per ton (24.933); alkalies, 115,744,091 lb., or 3.852 lb. per ton (3.560); and depressing reagents, 2,258,275 lb., or 0.166 lb. per ton (0.239).

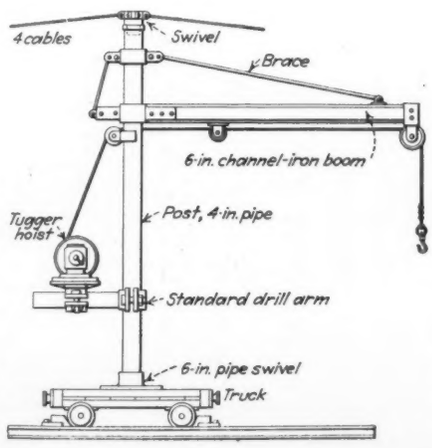
Abstracted from "Ore Concentration and Consumption of Flotation Reagents in 1931," Bulletin M.M.S. 174, U. S. Bureau of Mines, by T. H. Miller and R. L. Kidd.

Table II—Reagent Consumption in Treatment of All Ores

	Reagent Consumption: Total in 1,000 Lb.	Lb. per Ton Ore
I. Frothers:		
Pine oils	3,282	0.123
Cresylic acid	2,225	0.165
Orthotoluidine	1	0.008
Total frothers	5,508	0.153
II. Collectors:		
1. Distillation products:		
Coal-tar creosotes	491	0.186
Wood-tar creosotes	69	0.052
Pine-tar oils	5	0.006
Petroleum products	16	0.103
Blast-furnace oils	169	0.080
Water-gas tars	3	0.213
Totals	753	0.126
2. Synthetic products:		
Ethyl xanthates	2,411	0.111
Butyl xanthates	213	0.052
Amyl xanthates	301	0.028
Xanthate derivatives	65	0.011
Dicresol - dithiophosphoric acid	298	0.110
Sodium dicresol - dithiophosphate	106	0.035
Sodium diethyl - dithiophosphate	106	0.013
Thioamino phosphate	1	0.159
Thiocarbanilide	43	0.061
Totals	3,543	0.100
Total collectors	4,296	0.119
III. Acids and alkalies:		
A. Acids:		
Sulphuric	11,143	21.342
2. Alkalies:		
Sodium carbonate	878	0.586
Sodium bicarbonate	15	0.250
Sodium hydroxide	8	0.057
Lime	112,898	3.859
Cement	1,946	7.306
Total alkalies	115.744	3.852
IV. Other inorganic reagents:		
1. Sulphidizing:		
Sodium sulphide	643	0.723
2. Activating:		
Copper sulphate	3,325	0.593
3. Depressing:		
Cyanides	589	0.050
Sodium sulphite	368	1.158
Sodium silicate	262	0.862
Zinc sulphate	1,019	0.290
Sodium dichromate	14	0.350
Sodium sulphate	6	0.140
Total depressing	2,258	0.166
4. Miscellaneous:		
Sodium chloride	101	1.243
Sodium bisulphite	14	1.430
Sodium aluminate	1	0.040
Starch	36	0.252
Total reagents	143,068	3.979

Portable Timber Hoist

FLEXIBILITY in operation and freedom from injuries to the hands are obtained with the portable timber hoist illustrated. It was originally designed to eliminate the time-consuming hand methods employed in handling mine timber in the yards, writes F. P. Brunell, mining engineer, Shattuck-Denn Mining Corporation, at Bisbee, Ariz. Several novel features are embodied in its construction. All parts, consisting of a column made from 4-in. reinforced pipe, a detachable channel-iron boom with its guide pulleys and brace, and a sturdy four-wheel truck, were made from scrap material. The column rests in a swivel base mounted in the center of the truck, and the boom can be adjusted according to requirement by lossening the clamps on the boom and the brace—features which add materially to the utility of the hoist. Additional parts are a standard drill arm, a bolted-on hoist base, and a small IR tugger hoist. Four cables suspended from the clamp attached to the upper column swivel lend extra stability to the column.



Thin Sections of Weathered Rocks

MICROSCOPIC examination of rocks is becoming of greater and greater importance. Many rocks to be examined are altered or weathered, and the following notes on preparation of such rocks for examination have been made available through the courtesy of Dr. Thomas Clements, formerly with Cia. Minera de Peñoles, now assistant professor of geology at the University of Southern California.

The rock to be prepared is first broken (by hand, if soft enough) to a piece more or less cubic in shape and with faces about ½ in. square, one of which is then smoothed to a flat surface by gentle rubbing on a dry emery stone or on an emery wheel. A generous supply of Canada balsam (neutral) is heated on a glass slide until a tiny drop taken off on a pin point does not stick when touched to the thumbnail. With the slide still on the heater, the rock fragment is placed, smooth side down, in the balsam. Most weathered rocks are porous, so the hot balsam is absorbed immediately. The slide is quickly removed from the heater and allowed to cool.

When cool, the unimpregnated portion of the rock is cut or ground away, leaving a section two or three millimeters thick, firmly held together by the hardened balsam and ready for the final fine grinding and polishing. This is done in the usual manner—by grinding wet on a revolving lap (or by hand), first with comparatively coarse emery or alundum (No. 280), followed by No. 400, with a final polishing on plate glass with levigated alumina. The surface produced is ready for cementing on a new glass slide.

For this cementing, balsam is again heated on a slide to the same consistency; but this time the slide is removed from the heater and all bubbles are allowed to escape before the rock section is placed upon it, polished side down. An even, gentle pressure is maintained upon the two slides until the balsam has hardened, and then the first slide is removed by placing it on the heater and at the same time allowing alcohol to evaporate from the back of the second slide, thus keeping the latter cool, and causing the rock to continue to adhere to it.

The smoothed but unpolished surface that was first cut upon the rock section is now exposed, and this is cut and polished in the regular manner until the section has the standard thickness of 0.03 millimeter, as determined by the interference colors of quartz, feldspar, or whatever mineral may be present in comparatively large quantity. The section is covered by heating balsam to the proper point on a cover glass, allowing the bubbles to escape and then overturning it on the thin section, pressing very gently until the balsam has hardened. The rock section is now permanently mounted.

The following points should be noted:

1. The preliminary smoothing must be done dry; the rock is likely to go to pieces at this stage if wet. The pore space of the rock should not be filled with water when the balsam is introduced.

2. The balsam must be heated to just the correct consistency—no more, no less. When cooked too long it becomes brittle, and breaks away from the rock section, removing the all-important lateral support necessary to withstand the thrust of grinding. If undercooked, the balsam is sticky, and an attempt to reheat after the section is placed on the slide causes disintegration of the rock particles, with loss of original textures and structures.

3. With the exception of the first cementing, when it is desired to allow the hot balsam to impregnate as much of the rock as possible, the balsam should be removed from the heater and all bubbles allowed to escape before the section is placed in it. This reduces the number of bubbles between thin section and either slide or cover glass, even though it may not eliminate them entirely. Bubbles cannot be removed from thin sections of weathered rock as they can from hard-rock sections. In the latter, reheating may be resorted to and the bubbles squeezed out; reheating will ruin the weathered-rock section.

4. Abrasive coarser than No. 280 is likely to cause a tearing out of some of the mineral grains. The finer abrasive is somewhat slower, but time is saved in the long run.

5. The section must not be heated when the cover glass is cemented on. The latter should have the balsam evenly distributed over its entire surface; the cool rock section causes rapid chilling, preventing the balsam from spreading.

Control of Pulp Flow

AT THE new slime leaching plant of Inspiration Consolidated Copper Company, Inspiration, Ariz., difficulty was experienced in regulating the flow of slime from the storage tanks to the lead-lined agitators. The amount of pulp flowing into the agitators changed constantly, according to the pulp level obtaining in the storage tanks, and adversely affected acid consumption and general leaching operations. Means had to be found to provide an even flow of pulp, regardless of the ever changing pressure at the storage tanks. This was satisfactorily accomplished with the automatic control mechanism shown in the accompanying illustration, which was developed by Charles Warter, lead foreman. Fig. 1 presents the general arrangement of the installation, consisting of a small lead-lined tank; a sheet-iron float, with its guides made from standard pipe, secured to a wooden superstructure resting on the tank walls; and the control valve proper, built after the fashion of a check valve. Details of the design of the float and guides are shown in Fig. 2. The control valve, a cross-section of which appears in Fig. 3. is made from short pieces of scrap pipe, and is rubber lined to resist the abrasive action of the acid slime. Control levers, rollers, and guides are made from scrap material.

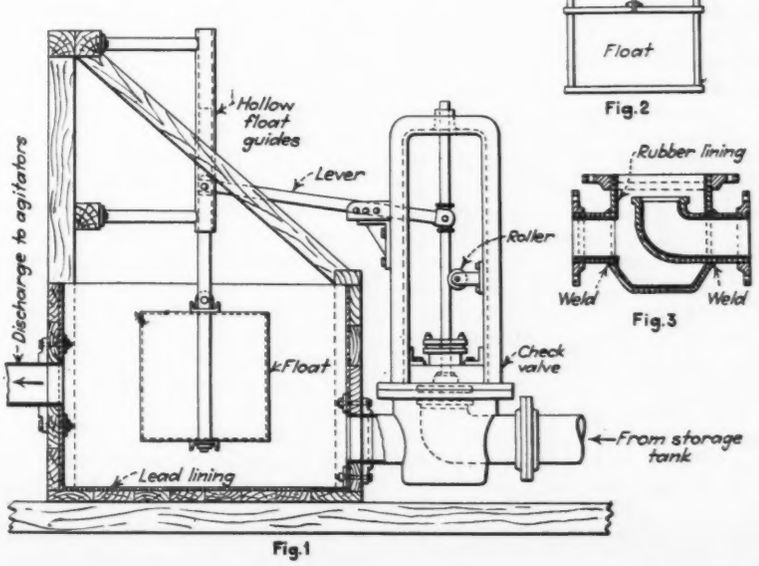


Fig. 1

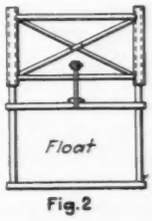


Fig. 2

Fig. 3

Scope for Gold-Ore Flotation

IN RECENT YEARS the flotation process has become more and more firmly established in the field of gold milling. It may be used to make a rough concentrate and a clean tailing at rather coarse sizes (48 mesh or so), thus eliminating much waste before finer grinding and further treatment of the concentrate; it may prove suitable to save small but worth-while amounts of base metals that would be lost in straight cyanide treatment; it may be of value to remove, in the form of a concentrate, refractory substances deleterious to cyanidation, producing a tailing that can be economically cyanided; it may be adopted for treating refractory tailing, such as the carbonaceous slime of the Mother Lode; or it may merely prove a more economical method of treating non-refractory ores. Flotation, in general, is applicable to ores in which the gold is associated with sulphide minerals or is of fine particle size and high grade, so that a stable froth can be maintained. Low-grade non-sulphide ores, according to Taggart ("Handbook of Ore Dressing," p. 866), are not ordinarily suited to the method, because there is not enough metallic substance present to stabilize a froth.

Cyanidation is most applicable to sulphide or non-sulphide ores that are free from excessive amounts of cyanicides. Tellurides require a preliminary roast or very fine grinding in conjunction with excessive aëration, or the use of bromocyanide or other special reagents. If considerable arsenic, antimony, or soluble copper, iron, or other cyanicides are present, the method becomes complicated and expensive. Cyanidation is best suited to the treatment of a large tonnage, because it requires a larger capital investment than do other methods.

If an ore is wholly or partly free-milling, and the gold is coarse, the first step after grinding is generally plate amalgamation, or recovery on riffle or blanket tables, unless organic substances in the water or other materials that promote sickening are present. Blanket recovery, followed by barrel amalgamation, is somewhat cheaper, uses less quicksilver, and is not so dependent on the skill of the operators as plate amalgamation. If cyanidation or other treatment follows, blankets may be preferable to plates. At the 70-ton mill of Granada Gold Mines, Rouyn, Que., Canada, muskeg water caused fouling of the plates, and blankets were substituted with satisfactory results. At the Sixteen to One mill, in California, riffled tables have wholly replaced plates in the ball-mill plant.

With ores requiring cyanidation a strong tendency to do away with prior amalgamation has prevailed in some districts until recently. The greatest possible simplicity in the flowsheet was sought, and much importance was attached to grinding in solution. A tendency in the reverse direction is now apparent. At the Dome mill, in Ontario, the original flowsheet involved amalgamation followed by cyanidation, but later the mill was changed over to an all-cyanide plant. It is now planned to return to the principle of recovering the coarse gold ahead of cyanide treatment, and blanket tables are to be installed. This is in accord with experience on the Witwatersrand.

When shaking tables are used after amalgamation and ahead of cyanidation, the concentrate, sand tailing, and slime tailing are sometimes treated separately by cyanidation. Good results are being obtained thus in California. An alternative procedure developed in Ontario employs Dorr bowl classification to promote selective grinding and concentration of the sulphides within the grinding circuits; this avoids need for separate treatment of concentrate, insures a simplified cyanide flowsheet, and reduces plant investment.

Most ores amenable to gravity concentration are also suitable for treatment by present-day flotation methods, which generally give superior metallurgical results at less cost; and a tendency is noticeable to substitute flotation for gravity concentration. Gravity methods are still the best when an appreciable amount of gold can be liberated from the gangue and concentrated after relatively coarse grinding.

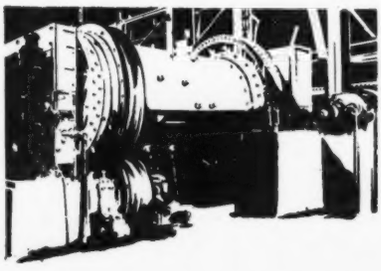
When the gold is finely disseminated in sulphides, very fine grinding is required for good cyanide extraction. Flotation may often make an equal or better recovery at much coarser sizes, because the sulphide particles can be floated when the gold is not exposed, as it must be for cyanidation. At the McIntyre Porcupine property, in Canada, most of the gold could be concentrated into 15 per cent of the original tonnage by bulk flotation at 48 or 65 mesh; the concentrate was then amenable to cyanide treatment after regrinding. A new mill has been built that avoids much grinding expense. Furthermore, the cyanide treatment is susceptible of accurate control, and much less plant space is required for a greater tonnage than was formerly handled in the all-cyanide mill.

A distinct advantage of flotation at some properties is the ability to recover valuable base metals. When arsenic or other cyanicides are present, flotation is a good substitute for cyanide. At Spring Hill, Mont., cyanide losses from arsenic and antimony led to the adoption of bulk flotation in 1929, with satisfactory results. Concentrate from this mill is smelted. Recent developments noted by the Mines Branch of Ontario, of the Canadian Department of Mines, are: Study of the possibility of flotation of ores containing submicroscopic gold so intimately locked in sulphides that cyanide extraction is unsatisfactory; recognition of the fact that, inasmuch as 70 to 80 per cent of the gold in many ores treated by cyanidation is dissolved in the grinding circuit, with most of the remaining gold extracted on the filters, further filtration might be substituted economically for the long agitation period; and progress in improving and applying methods for the regeneration of cyanide.

Many considerations enter into the choice of methods or combinations of methods. Thus, straight amalgamation or blanket treatment may be applied in small plants; amalgamation and concentration (by gravity, flotation, or both), followed by cyanidation of concentrate and tailing, or smelting of concentrate and cyaniding of tailing; cyanidation followed by flotation of tailing; flotation followed by smelting or cyanidation of concentrate and discard of tailing; flotation followed by cyanidation of concentrate or tailing, or both; and so on.

From the standpoint of cost, blanket recovery and amalgamation are cheapest to operate and to install. Flotation is cheaper to operate, and requires a less expensive plant, than cyanidation. Gravity concentration at relatively coarse sizes requires a less expensive plant than does flotation. Gravity concentration and flotation involves greater plant investment than does straight flotation; but an amalgamation-vanner concentration plant, as on the Mother Lode, is somewhat cheaper than an all-flotation mill. An all-cyanide slime plant usually costs nearly twice as much as does a straight flotation mill, per ton of capacity. Many exceptions occur, but the foregoing generalizations give a broad picture of the relative plant investment needed for different milling methods.

Sound judgment based on long experience is necessary in the proper design of flowsheets. Huge sums have been wasted because of the failure of small companies to recognize this fact and to entrust the technical problems of milling their ores to men capable of solving them correctly.



Condensed abstract from Bulletin 363, By C. F. Jackson and J. B. Knaehel, U. S. Bureau of Mines.

Cost of Mine Transportation

Morris J. Elsing

THE PROCESSES involved in mine transportation are variable. The amount of material to be transported usually determines or decides the character of each operation. In the instance of a small mine with a vertical shaft, a cage is almost always included in the hoisting equipment. The advantages of skip hoisting can be secured, even in a single-compartment shaft, by using a skip, either above or below the cage. Ore and waste are hand trammed in cars to the shaft. With the combined cage-skip, ore or waste can be moved economically, and to any destination required. The run-of-mine ore can be hoisted in the skip; special ore can be hoisted in cars on the cage; and, similarly, waste can be hoisted in cars to the surface, or to levels where it may be needed as filling. In most inclined shafts of metal mines, hoisting is done in skips. In the smaller operations of skip hoisting, in vertical or inclined shafts, the method of skip loading varies. Some conditions make more desirable the loading of the skip by dumping the car directly into it; under other conditions, to provide small pockets from which the skip is loaded has been found more satisfactory. As operations increase in size, larger capital outlays are warranted, so that costs may be reduced. In the smaller mining operations, where tramming distances are considerable, animal haulage is adopted, and mules are more commonly used than horses. In most mines, ore is transported from the chute or working face in cars. The general exception to this practice is found in the lead-zinc districts of Missouri, Kansas, and Oklahoma, where buckets or cans, holding over 1,200 lb. are set on trucks and trammed to the shaft, to be hoisted and dumped.

As the mine output increases, mechanical devices replace hand labor, and more elaborate and larger equipment is added to the transportation system. Electric trolley or storage-battery locomotives are then the commoner forms of motive power, although compressed-air locomotives are occasionally used, and, more rarely, gasoline locomotives. Instances of the use of belt conveyors and of rope haulage in metal mines in the United States are exceptional.

Mechanization is carried to the highest possible degree in the larger underground transportation systems. The more important loading chutes are equipped with compressed-air operated doors or gates; trains are run with automatic block signaling systems; the doors of air locks on the haulageway are automatically opened and closed; cars are unloaded at the shaft into ore pockets by mechanical dumping devices, of which the tipple is the most common. In the largest operations, hoisting has become entirely mechanical, attendants rather than operators being required.

In a series of papers to follow, the cost of every means of surface and underground metal-mine transportation will be discussed. Cost data to be presented have been accumulated by extensive search of the literature on the subject. Of the mass of data collected, the most important costs are considered to be those that can be presented on a comparable basis. In the larger operations, use will be made of a per ton-mile basis of comparison; in the smaller operations, the length of haul will be used. Methods of keeping costs at different mines vary considerably, and much difficulty has been found in presenting costs in a strictly comparable form. In some instances distinctly different operations overlap, and no definite cut-off is possible. The operation of mucking and tramming loaded cars is often performed by the same man, and no definite cost segregations are made between them. In showing costs, as far as possible, all such conditions will be noted.

Cost per ton for transportation is clearly a function of the type and character of the orebody being mined. The method of mining is also determined by the physical conditions under which the ore occurs. A logical procedure, therefore, is to segregate total costs of transportation per ton for each method of mining. This segregation has been made in Table I. These costs, representative of 66 mines, apply to the transportation of 44 million tons of ore.

In these costs, transportation includes all of the processes involved in the delivery of ore from the mine chutes to its arrival at the collar of the shaft. It, therefore, includes the loading of mine cars from the ore chutes, the hand or mechanical tramming of these cars to the shaft, caging, hoisting, and dumping. No uniformity exists as to how charges for transporting waste rock, timber, tools, and other supplies are made. To illustrate: waste-rock transportation might be charged to development, or even to stoping when the waste is required for fill. In some instances, the timber transportation charge is added to the cost of timber. In other instances all transportation is charged to one account, and is then proportioned on the basis of the tonnage mined.

As a rule, transportation charges do not include depreciation or amortization of the equipment, but general repairs are included in the costs.

The simplest of all methods of transportation is man-packing. This method never was employed in the United States, but it is still used in several of the cheap-labor countries. Chicken ladders and rawhide sacks are the only equipment needed. Because of the exceedingly low wages paid, costs by this method of transportation are proportionately low.

Next in point of simplicity is hand tramming, by far the most common. It is used exclusively in the majority of mines; and in a great many more it is an important auxiliary to mechanical tramming. General practice tends towards the use of light cars run over light rails of narrow-gage track. For the most economical hand tramming, cars should have as great a capacity as one man can readily handle. Capacity depends principally upon the type of

Table I—General Transportation Costs

Mining Method	Number of Mines	Average Transportation Cost per Ton	Per Cent of Total Mining Cost	Tonnage
Caving				
Block	4	$0.120	16.1	15,097,798
Sublevel	2	0.335	26.4	950,000
Open stope				
Tri-State District	5	0.178	22.2	863,556
Wisconsin	4	0.172	24.6	.....
Michigan Copper	2	0.465	36.4	1,943,518
Iron mines	2	0.423	34.6	1,240,000
Glory hole	3	0.389	35.9	850,135
Sublevel stoping	5	0.210	17.7	1,576,615
Top slicing				
Michigan	3	0.140	13.0	998,700
Western United States	3	0.150	8.4	.....
Shrinkage				
Narrow veins	5	0.556	16.9	388,500
Wide veins	5	0.369	23.0	1,278,084
Large orebodies	3	0.109	22.8	14,187,412
Cut and fill				
United States	7	0.468	12.9	2,317,102
American, foreign	4	0.306	10.5	1,246,224
Square-set				
All mines	9	0.508	10.4	1,243,458
Totals	66			44,181,102

One of a series of articles on the cost of ore transport in mines.

running gear of the car and the character of the track. Cars should be provided with roller bearings, and track conditions should be as nearly prefect as possible. Under such circumstances one man can easily handle a 20-cu.ft. car with a capacity of a ton or slightly more. The most common grade is 0.5 per cent or 6 in. per 100 ft. in favor of the load. Turn sheets, except in rare instances, have become obsolete. Good track switches are as important a part in any well-regulated hand tramming system as they are in mechanical haulage.

The cost per ton of tramming and a brief summary of conditions under which it is done are given for a number of widely scattered mining districts.

Mine No. 3, Tri-State District, I.C. 6174—Muckers fill cans, each with a capacity of 0.6 ton of ore. Trammers tram these cans on trucks to the shaft, but do not unload them. In 1927, 132,384 tons was trammed, with an output per trammer-shift of 57.25 tons. Average distance trammed was slightly less than 400 ft. With trammers' wages at $3.50 per day, the cost, including only tramming, was 6.1c. per ton. The above wage is based on $40 zinc ore. For each $5 increase in price, wages increased 25c. per shift.

Mine No. 1, Marquette Range, Michigan, I.C. 6138—Ore cars have a capacity of 2.8 short tons of iron ore. Trammers work in pairs on contract and are paid 20c. per car, or 7.1c. per ton, for loading from chutes, and 1c. per car, or 0.36c. per ton, for each 45 ft. of the tram. On this basis the cost for loading from chutes and tramming 500 ft. is 11.1c. per short ton.

Cleveland-Cliffs, Ishpeming, Michigan, "Min. & Met.," March, 1927—Cars have a capacity of 2.8 short tons and were trammed by two men a distance of 1,500 ft. Fifteen cars or 42 short tons were trammed per shift. Trammers received $4.35 per shift; and, on this basis, the cost for tramming 1,500 ft. was 20.7c. per short ton. For the same tram the cost by storage-battery locomotives was 9c. per short ton.

Elko Prince, Midas, Nevada, A.I.M.E., Vol. 60—Two men trammed 60 tons per day and loaded directly into a skip. The average tram was about 200 ft. The cost, including loading from chutes, tramming, and dumping, averaged approximately 18c. per ton.

Smuggler-Union, Telluride, Colorado "E. & M. J.," Oct. 23, 1930—Table II shows the details of the cost of tramming 16,301 tons by contract. The average distance was 283 ft. The shrinkage method of mining was used, which accounts for the high cost per ton.

Black Rock, Butte, Montana, I.C. 6370—On most of the levels, Anaconda-type, end-dump, 13-cu.ft. cars are used. Maintenance of long drifts in heavy ground makes the use of larger cars impracticable on most levels. The track gage is 18 in. Tramming is done on contract at prices that vary from 1.5c. to 3.0c. per cubic foot of ore in place. The estimated cost for an average 500-ft. tram varies from 15c. to 30c. per ton.

Table II—Hand Tramming By Contract March to Aug. 1927

Charge	Cost Per Ton
Contract labor	$0.215
Ammunition	.029
Car repairs and supplies	.023
General tramming overhead	.014
Nipper labor	.008
Total	$0.289

Utah Copper Co., Bingham Canyon, Utah, A.I.M.E. Vol. 54—When ore was mined by underground mining methods it was trammed on the two upper levels from stope chutes to transfer raises. The maximum tram was 150 ft., and the average distance was about 90 ft. Cars held one ton of porphyry ore, and two trammers were used on a car. The average tonnage trammed by a crew was 65 tons per shift, including loading, tramming, and dumping. The cost of tramming (estimated) was 10c. per ton.

Verde Central, Jerome, Arizona, I.C. 6464—The end-dump type of car with roller bearings was used. Cars had a capacity of 16 cu.ft. and held about 0.8 ton. The track was of 16-lb. rails and the gage was 18 in. Trammers loaded their own cars and switched them at the shaft station. The average number of cars trammed per man-shift was 23, and the average distance was about 500 ft. The tramming wage was $5.06 per day. On this basis the 500-ft. tram cost 27.5c. per ton. The principal method of stoping was by shrinkage. The ore was fairly hard. Loading under these conditions accounts for the high tramming cost.

United Verde, Jerome, Arizona, A.I.M.E., Vol. 66—Ore from the stopes on the 1,650 level was trammed to transfer raises, an average distance of 200 ft. An average of 44 tons per shift was handled by hand trammers at a cost of 12.7c. with trammers' wages at $5.60 per shift. In tramming waste to the stopes, an average distance of 350 ft. up a 0.4 per cent grade, the cost of loading, tramming, and dumping was 22.4c. per ton. Twenty-five cars were trammed per shift.

Ground Hog, Vanadium, New Mexico, I.C. 6377—Tramming and hoisting are done on a bonus system. The tram boss loads the cars and has charge of the tramming. His daily rate is $4.25, plus 1c. per car hoisted. Two trammers are paid $3 per shift, plus 2c. per car hoisted. The cager receives $3.50 per shift, plus 2c. for each car, and the top lander $3.50, plus 1c. per car hoisted. The average tramming distance is about 700 ft. and each car contains 0.9 ton of ore. An average of 90 cars or 81 tons is hoisted per shift. On this basis the 700-ft. tram, including loading from chutes but not caging and hoisting, costs 18.2c. per ton.

New Jersey Zinc Co., Franklin, New Jersey, A.I.M.E., Vol. 57—The cars used for hand tramming were of 24-cu. ft. capacity and were intended to be handled by two men. They weighed about 2,000 lb. and held 1.4 tons of ore. The track gage was 24 in. and the rails weighed 20 lb. per yard. Trammers loaded from chutes, trammed, and dumped cars. Trammers' wages were $1.85 per shift. On this basis, in 1916, tramming cost 11.5c. per ton for a tramming distance of 250 ft. These costs are based on the tramming of 552,625 tons.

85 Mine, Lordsburg, New Mexico, I.C. 6413—All ore is hand-trammed in 16-cu.ft., 1-ton, roller-bearing, end-dump cars. The average distance trammed is 1,200 ft. The track grade is 0.5 per cent in favor of the load, and the gage is 18 in.; 16-lb. rails are used.

Tramming costs in July and August, 1930, were 67.0c. per ton, when 14,294 tons of ore was trammed. Of this amount, 8.3c. was for supplies. Trammers' wages were $3.60 per day. The mine was worked by the shrinkage method, and chute loading with bulldozing under such conditions, when charged to tramming, greatly increased the expense. The tramming cost includes tramming ore and waste, chute blasting, and supplies, such as mine cars and oil.

Cananea, Sonora, Mexico, I.C. 6247—All hand tramming is done with 20-cu. ft., 1-ton, end-dump cars weighing 1,060 lb. These are run on 18-in. gage tracks of 16-lb. rails laid on 4x6-in. ties, 4 ft. long, spaced 2 ft. center to center. Trammers are paid on contract for loading from chutes and tramming to the shaft. Where the average tram is about 800 ft., the average performance is 25 cars per shift. A flat rate of 20 to 25 centavos is paid per one-ton car. On the basis of exchange at two for one, the cost of tramming is 10c. to 12c. per ton.

Lucky Tiger, Sonora, Mexico, A.I.M.E., Vol. 72—Most of the tramming is done by contract. Cars are rotary-dump of 16-cu.ft. capacity and hold ¾ ton. The contract price for distances up to 500 ft. is 5c. per car, or 6.7c. per ton. For distances over 500 ft., the price is 7.5c. per car or 10c. per ton. When the cars are loaded by shovel, the price is 12.5c. per car or 16.7c. per ton.

Miscellaneous older data, A.I.M.E., Vol. 49—Carl A. Allen gives data relative to tramming at the Erie Consolidated, Calif.; Pittsburg-Silver Peak, Nev.; Ohio Copper, Utah; and Cananea, Son., Mexico. These data apply to conditions in 1911 and 1912. They have been recalculated and are shown in Table III.

Peele's "Mining Engineer's Handbook" states that "tramming duty with 18-cu.ft. cars, on 18-in. gage over average mine track, varies from 0.8 to 1.4 minutes per ton per 100 ft. or approxi-

Table III—Direct-Labor Cost per Ton of Hand Tramming

	Size of Car, Tons, Capacity	Tons per Man-Shift	Average Tram, Ft.	Cents per Ton	Remarks[1]
No. 3 Tri-State	0.6	57.0	350	6.1	T only.
No. 1 Marquette	2.8	....	500	11.1	L, T.
Cleveland-Cliffs	2.8	21.0	1,500	20.7	L, T.
Elko Princes	...	30.0	200	18.0	L, T, D.
Smuggler Union	...	....	283	21.5	L, T, Contract.
Black Rock	0.7	....	500	22.5	L, T, Contract.
Utah Copper	1.0	32.0	90	10.0	L, T, D.
Verde Central	0.8	23.0	500	27.5	L, T.
United Verde	1.0	44.0	200	12.7	L, T, D.
United Verde	0.9	25.0	350	22.4	L, T, D, Upgrade.
Ground Hog	0.9	27.0	700	18.2	L, T, Bonus.
New Jersey Zinc	1.4	16.0	250	11.5	L, T.
Colorada, Mex	1.0	25.0	800	11.0	L, T.
Lucky Tiger, Mex	0.8	....	350	6.7	L, T, Contract.
Lucky Tiger, Mex	0.8	....	600	10.0	L, T, Contract.
Older Data					
Erie Consolidated	1.2	25.0	1,500	12.0	L, T.
Pittsburg Silver Pk	1.1	50.0	700	7.0	L, T.
Ohio Copper	1.0	82.0	150	4.0	L, T.
Cananea, Mex	0.8	34.0	300	4.4	L, T.
M. E. H.[2]	0.9	29.0	1,000	....	L, T.
M. E. H.	0.9	81.0	100	....	L, T.
Copper Country[3]	2.0	14.8	609	22.8	.....

[1] L indicates load from chutes; T, tram; D, dump.
[2] "Mining Engineer's Handbook."
[3] Most of this ore was loaded by shoveling.

mately one useful ton-mile per hour minus loading and dumping. For loading from chutes and delivering to a shaft station, under best conditions, the average delays at terminals through the shift will be not less than 4 min. per round trip. On this basis, with an 1,800-lb. load, one man can tram 32 cars 1,000 ft. or 90 cars 100 ft. in 7.5 hours' actual work, and his output will be 5.5 and 1.7 useful ton-miles, respectively."

Michigan Copper Country, "E.&M.J.," Nov. 21, 1914—Interesting costs from the Copper Country, in Table IV, are not exactly comparable with the preceding costs. In the first place, most of this tonnage was loaded into cars by shoveling. Besides, wages were much less during this period. For the first six months of 1913, trammers averaged $3.17 per shift and contract-trammers $3.40. Cars were open-end, with a capacity of 40 cu.ft., or about 2 tons, and were handled by two men.

Three processes are usually included in hand tramming—loading, tramming, and dumping. Of the three, under usual conditions, loading of the car is the most important as effecting costs. The cost of this part of the operation varies much more than do the actual costs of tramming or dumping of the material trammed. Several factors determine the cost per ton, of which the most important is the character of the material handled. With a free-flowing material, low loading costs can be obtained with almost any type of chute gate, and the size and type of car are not of such importance. With material that is clayey, sticky, and damp, the difficulties of loading are innumerable and often even dangerous. Under such conditions, the material has to be barred down or dug out of the chutes; it hangs up in the chute raise; and, when it does move, it falls with such violence that damage is often done to the chute mouth or raise lining. The method of stoping has an important bearing on the character of material drawn from chutes. With the shrinkage method, the ore produced usually contains many large pieces. Where large boulders are numerous, the item of bulldozing is an important part of the total cost of mining. The highest costs shown in Table III are, in nearly every instance, at mines where the shrinkage method is used. These costs would be materially increased if the cost of powder and chute repairs, a part of the total loading cost, were included. With the square-set, cut-and-fill, and top-slice methods, the top of the chutes are provided with grizzles, to prevent entrance of large pieces of ore that would hinder loading.

Loading is an important part of tramming, so the tram length does not usually have a very important effect on cost. Technically speaking, length of tram does not even have a proportional effect upon the cost of tramming if considered apart from loading and dumping. The greatest physical exertion is required and the most time is consumed in starting and stopping the car. Attempt to try to determine the effect of this, for varying lengths of haul, would be impracticable. Suffice to say, the cost of the single operation of tramming is approximately proportional to the distance.

Available data indicate that a trammer will tram a 1-ton car at 200 ft. per minute. When allowances are made for the time required for the trammer to get from the collar of the shaft to his working place, and to return, and for his lunch hour, the working time of the 8-hour shift is reduced, so that the distance trammed per minute of the shift is only 160 ft. On this basis, the tram with the load is 80 ft. per minute of the total shift. Tramming at this rate can only be done when good cars are provided and when the track and roadbed are kept in perfect condition. Of the other factors that go to make up an efficient hand tramming system, probably none is so important as that there should be an ample number of cars.

The satisfactory use of the 1-ton car is impossible with some ground conditions, as in the instance of very heavy ground where the timbers are under great pressure and are constantly breaking. When this condition exists the size of drifts may be advantageously decreased, which necessitates the use of small cars. Among the mines in Table III, this situation prevails at the Black Rock, where cars have a capacity of only 13 cu.ft. Obviously the use of smaller cars increases costs. Cost of loading cars under many conditions is almost independent of size. A car is spotted, and when the stream of ore is once started, double the tonnage drawn does not require double the effort. The actual tramming and dumping costs are also independent of the size of the car. The small size of cars at the Black Rock mine accounts for the comparatively high cost.

To complete the discussion: adverse track grades will easily double the cost of actual tramming. This condition is more likely to affect the cost of moving waste filling than ore. Haulage drifts when originally driven are always designed to benefit the transportation of ore, whereas waste must often be moved under opposite conditions.

What has been said regarding the drawing of sticky ore from chutes also applies to the dumping of mine cars containing this ore. The cost of car dumping is, therefore, also a variable figure.

The data available indicate that the cost of loading from chutes, under normal conditions, will average about 7c. per ton, that tramming with 1-ton cars will average about 1c. per 100 ft., and that dumping will average about 2c. per ton. Under some conditions of shrinkage mining the loading cost may be more than double this amount. When these factors are applied to the costs shown, they will be found to have a much closer relation than is apparent from the data that are presented in Table III.

In this discussion of costs the direct-labor cost only has been considered. Indirect costs of track maintenance and drift repairs are so variable that estimates of averages would be of little value. Under good ground conditions, the indirect costs for supervision, chute, track, and drift maintenance and repairs, and car maintenance and repairs will increase costs by about 50 per cent of the direct-labor cost.

Table IV—Michigan Copper Costs, Jan. 1 to June 30, 1913

	Average Tram, Ft.	Tons per Man-Shift	Estimated Cost per Ton, Cents
Ahmeek	563	16.1	20.5
Allouez	753	12.9	25.7
C.&H. Osceola	582	19.5	16.9
C.&H. Conglomerate	171	14.5	22.7
Centennial	900	14.0	23.6
Isle Royale	750	14.1	23.3
North Kearsarge	702	13.4	24.7
South Kearsarge	444	14.8	22.3
Osceola	850	12.7	25.9
Superior	480	18.3	18.1
Tamarack	508	12.1	27.2
Average	609	14.8	22.8

Table V—Mule Haulage Data

	Mine No. 1 (a)	Mine No. 2 (b)	Mine No. 3 (c)	Mine No. 4 (d)
Average tons trammed per shift	180	288	400	256
Average tram distance, ft.	1450	2613	2006	1438
Number of mules	4	9	3	2
Mineral tons per mule-shift	45	32	64	128
Miles traveled per mule-shift	12	15.8	8.5	18.1
Ton-miles per mule	12	15.8	26	36.3
Cost per ton, cents	7.9	10.9	5.2	2.8
Cost per ton-mile, cents	30.0	22.4	14.0	10.1

(*a*) Grades against mules, 300 ft. at 3 per cent.
(*b*) Mules pulled two cars to layby station, 1,000 ft. from shaft, grades adverse, 1 per cent to layby; 8-car trains from there to shaft.
(*c*) Level grades, 6-car trains, traveled 3.5 miles per hour.
(*d*) Excellent conditions, 1.5 per cent down grade, travel rate 5 miles per hour, practically no pull; 4 empty cars are limit that can be returned against this grade.

As a factor in underground transportation, animal tramming is constantly becoming of less importance. The method is mainly used in the lead-zinc districts of Central United States; and, even there, it is being gradually replaced by trolley and storage-battery locomotive haulage.

A review of mining literature on the subject of animal transportation underground gives the following information relating to its economic aspects:

Southeast Missouri, A.I.M.E. Vol. 48—H. A. Guess gives average tramming costs underground for gathering with mules and haulage to the shaft as 5c. per ton, exclusive of track work.

Acme, Waco, Tri-State District, I.C. 6150—Trains of five or six loaded cans mounted on trucks, each containing 0.6 ton of ore, are hauled by mules to the shaft "layby" and the empties returned to a point near the shovelers in the working faces. Main haulage tracks consist of 12-lb. rails with 18-in. gage. Average tramming distance is 500 ft. Average mule haulage cost in 1928 was 6.4c. per ton, including supervision and all supplies.

Joplin District, A.I.M.E., Vol. 57—H. I. Young says a mule will haul 100 tons 1,500 ft. in an 8-hour shift at a cost of 5 to 6c. per ton. In four months of 1917, 81,145 tons was hauled an average distance of 700 ft. Details of costs in cents per ton are: Mule driver, 2.3; maintenance mules, 0.5; car couples, 1.2; car greases, 0.6; total, 4.6c. per ton. On the basis of these figures, mule haulage cost 35c. per ton-mile.

Wisconsin Zinc District, A.I.M.E., Vol. 63—Boericke and Garnett say that mine cars have a capacity of 16.5 cu. ft., or 0.9 ton. The tracks are of 16-lb. rails laid at a 22-in. gage. A mule that pulls two cars for a distance of 1,200 ft. can handle dirt from one shoveler. A mule will make from 12 to 20 ton-miles per shift at a cost of 25c. to 30c. per ton-mile.

Mule Haulage in Metal Mines, "E.&M.J.," Nov. 26, 1921—Table V gives some very interesting data relative to mule haulage under varying conditions.

Anaconda, Butte, Mont., A.I.M.E., Vol. 68—C. D. Woodward says: "There are now (1922) operating in the Butte mines of Anaconda Copper 185 trolley-type locomotives, each of which has replaced five animals. It cost approximately $50 per month to keep an animal; hence the economies that resulted fully justified the expenditure of electrifying the haulage system."

Anaconda, Butte, Mont., A.I.M.E., Vol. 68—According to H. R. Turnell, ten cars (18-cu.ft. capacity each) formed the standard trip for one horse. The cost of a horse per day was as follows: Shoeing, $3 per month, $0.10; stable boss, caring for 12 horses, 0.50; feed, 0.50; total cost per day, $1.10.

Elko Prince, Midas, Nevada, A.I.M.E., Vol. 60—Dorr and Dougan state that mine cars were 30 cu.ft. capacity, Truax side-dump type, weighing 1,100 lb. One man and a horse handled 60 tons a day in 5-car trains with an average haul of 1,100 ft. The haul was from the shaft pockets on the 300 adit level to the mill bins. On the basis of these figures, with the average trammer wage and animal cost, the cost of transportation was 10.8c. per ton, or 52c. per ton-mile, and the output was 12.5 ton-miles per mule-shift.

Jardine, Jardine, Montana, "E.&M.J.,"

Table VI—Average Costs of Animal Transportation

	Length of Haul, Ft.	Ton-Miles per Mule-Shift	Cost per Ton, Cents	Cost per Ton-Mile, Cents
Southeast Missouri	.....	....	5.0	....
Acme, Tri-State	500	....	6.4	....
Joplin District	700	....	4.6	35.0
Wisconsin	.....	16.0	....	27.5
Mine No. 1	1,450	12.0	7.9	30.0
Mine No. 2	2,613	15.8	10.9	22.4
Mine No. 3	2,006	26.0	5.2	14.0
Mine No. 4	1,483	36.3	2.8	10.1
Elko Prince	1,100	12.5	10.8	52.0
Jardine	2,250	....	10.5	25.0
Average	1,625	19.8	7.2	27.0

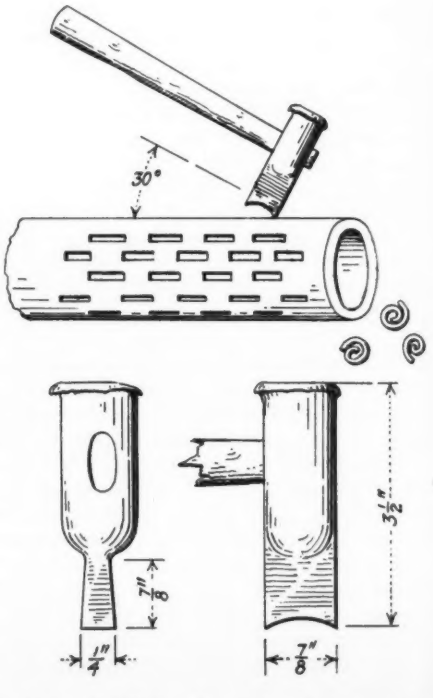
Nov. 14, 1925—Mule tramming through an adit tunnel, a distance of 2,250 ft., cost 10.5c. per ton, or at the rate of 25c. per ton-mile. A summary of the data above presented is shown in Table VI. Figures in this table, in nearly every instance, do not include loading and dumping. Possible exception to this statement may be the Elko Prince and Jardine mines. Based on the data presented, cost of transportation by animal tramming varies from 25c. to 30c. per ton-mile, and the output per animal varies from 12 to 26 ton-miles per shift.

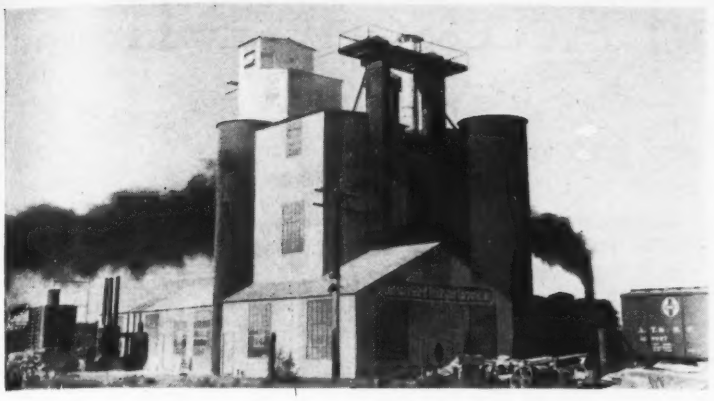
Pipe Perforating Tool

OCCASIONALLY, the mine shop is confronted with the problem of perforating heavy pipe, writes Evan W. Baker, from Winkleman, Ariz. This job is ordinarily done by drilling, which is slow, because of difficulties in handling a long and heavy piece of pipe on the drill-press table. The tool described here does a much speedier job just as neatly. With this simple cutter a pipe can be divided lengthwise in a few minutes.

The cutter is forged from carbon tool steel, an eye being formed in the head to take a hardwood handle. Relief is given to the cutting edge by tapering the bit slightly, making the cutting edge thicker than the metal directly behind it. This prevents binding in the slots the tool cuts. The concave shape is given to the bit by filing. Finally, the bit is tempered, the temper being drawn to a light purple, and not more than $\frac{1}{4}$ in. behind the cutting edges.

Operation of the cutter is simple. One man holds the tool while another does the striking with an 8- or 10-lb. hammer. The bit should be held at an angle of about 30 deg. to the work. The shaving, formed as the cutter advances, may be broken off by lowering the handle and striking the head of the tool a sharp blow. Dimensions, as shown in the sketch, are suitable for pipe up to 8-in. diameter. For larger pipe a heavier tool should be made.





Feldspar and Silica Processing

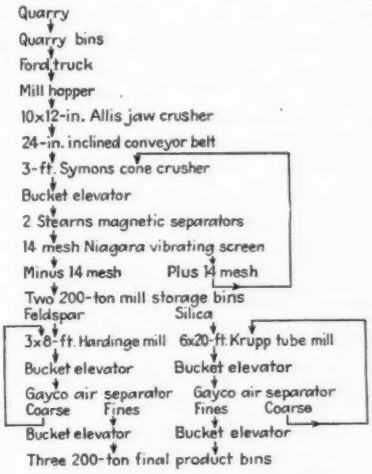
A. J. McDermid

THE Kingman mill of Consolidated Feldspar, a company with headquarters in Trenton, N. J., is 2 miles east of Kingman, Ariz. It has been in operation for about a year and is now being run on two shifts. Plant capacity per 24 hours is 100 tons of feldspar or silica to minus 20 mesh, or 35 tons to minus 200 mesh. These minerals are obtained from the corporation's quarry in the mountains, 6 miles north of the mill. The chief product is the minus 20 mesh feldspar, sold to the glass industry. Both silica and feldspar are ground to minus 200 mesh for manufacturers of pottery. A typical percentage analysis of the feldspar, which is orthoclase, is as follows: silica, 65.16; alumina, 19.32; iron oxide, 0.06; calcium oxide, 0.26; magnesia, 0.09; soda, 2.48; potash, 12.18.

With the present flowsheet only one kind of material can be treated at a time. The rock, broken to 10 in. in the quarry, is hauled to the mill in a Ford truck. After being weighed on a 15-ft. Howe platform scale, it is dumped into a hopper, from which it is fed to a 10x12-in. E. P. Allis jaw crusher. The crusher reduces the size to 1 in. and discharges on to a 24-in. inclined belt conveyor, which feeds a 3-ft. Symons cone crusher, where the rock is reduced to minus 14 mesh. The crusher product is elevated by a bucket elevator to two Stearns magnetic separators, and then passes over a 14-mesh Niagara vibrating screen. The plus 14 mesh material from the screen goes by gravity back to the cone crusher, and the minus 14 mesh material drops into two 200-ton capacity, 12x51-ft. circular steel storage bins through a spout into a 3x8-ft. Angeles.

If feldspar is the product being ground, it is fed from the mill-storage bins through a spout into a 3x8 ft. Hardinge mill; if silica is being ground it is fed in the same manner to a 6x20-ft. Krupp tube mill. French pebbles are the grinding medium in both mills, which are lined with silex blocks. The mills are each in closed circuit with a Gayco air separator, which can be set to give a product of any desired fineness. Fines from the separators are carried by bucket elevators to three final-product bins of the same type and size as the mill-storage bins. The final product can be dropped directly into box cars from the bins or can be drawn off into sacks.

Power for the plant is furnished by a 185-hp. Victor diesel engine. Power is transferred from the diesel by belts to a 90-kw. G. E. generator and to a shaft, which is connected to the cone crusher by a belt and to the Hardinge



Flowsheet of the Kingman plant of the Consolidated Feldspar Corporation

and Krupp mills by gears. The generator furnishes electricity for a 30-hp. Westinghouse alternating-current motor driving the jaw crusher, for several smaller elevator and conveyor belt motors, and for lights. The elevator motors are connected to the bucket elevators by Pacific Gear & Tool motorized gear reducers. Fuel consumption for the diesel engine is 7.5 gal. of 27-plus gravity fuel oil per hour.

The mill crew on each shift consists of a mechanic, a millman, a truck driver, and a laborer. Most of the quarried material is stored in bins at the quarry and is sent down by truck as it is needed at the mill, but emergency storage piles of both silica and feldspar are kept in the mill yard.

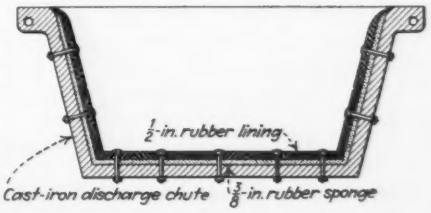
Feldspar is quarried from a 25-ft. vertical feldspar dike intruded into the surrounding granitic rocks. Silica is obtained from a 1-ft. quartz vein that traverses the feldspar dike parallel to its strike. The equipment at the quarry consists of a Denver Rock Drill stoper and plugger and an Ingersoll-Rand 9x8-in. single-stage compressor driven by a 25-hp. Western semi-diesel engine.

Little drilling and blasting is necessary, as the rock is not hard. The explosive used is Hercules 40 per cent ammonia powder. Detonation is accomplished electrically. The rock is mucked and trammed by hand from the quarry to wooden bins, from which it is drawn into the truck to be hauled to the mill. The quarry crew consists of a miner and four muckers, who work day shift only.

D. E. Walsh, of Kingman, is superintendent in charge of operations at both mill and quarry, and to him acknowledgment is due for the information contained in this article.

Rubber as a Chute Lining

APPLICATION of rubber lining on the wet charge and discharge chutes on the rock washers at the plants of New York Traprock Corporation near Newburgh, N. Y., has effected significant economies. Contrary to the common practice, Roy Coy, master mechanic, uses a $\frac{3}{8}$-in. layer of Goodrich rubber sponge between the cast-iron chute and the rubber lining proper, to act as a cushion and prevent fine rock particles from coming in contact with the chute bottom, resulting in increased life of the lining. Both lining and sponge are held to the chute walls by standard, flat-head, $\frac{1}{2}$-in. bolts. The life of rubber lining is about four times greater than that of any kind of steel lining.



Rock Drills and Accessories—II

Lucien Eaton

OPINION differs as to the benefits of "normalizing" fatigued steel—heating it to about 1,400 deg. F. and allowing it to cool slowly. This is done sometimes in a furnace and sometimes by passing an electric current through it, as in an electric rivet heater. Normalizing relieves "fatigue" of the metal, but removes the effects of a low finishing temperature in rolling, and danger is always incurred of bending the hot bars. The fact is not generally known that "fatigue" and internal stresses in steel can be relieved to a large extent by rest at ordinary temperatures. If steel that has become brittle by use is stored for six months in a shed, where it is subject to the natural daily variations in temperature, it will recover much of its original toughness.

The only method of testing steel for durability in advance is what is known as the "block test." A standard drifter is mounted rigidly on a bar, and a 4- to 5-ft. length of steel, squared at both ends, is driven by this machine against a heavy steel block, until it breaks. The time that elapses before failure occurs is measured with a stop watch. By thus testing samples of steel of the same length and section under identical conditions, a good comparison may be obtained of the relative durability of different grades and makes of steel. The test is not quantitative—one cannot predict from it the life of the steel in service, because of the difference in working conditions—but it does serve as a basis of comparison.

The amount of steel required per drill per year depends on the hardness of the ground, the quality of the steel and its size, and the conditions under which it is used. Hard ground, high air pressure and use of small steel increase incidence to breakage; care in sharpening, tempering, and handling decrease it. On the average, the consumption per machine per shift per year is 400 to 500 lb. Bits dulled will average about one for every two tons of ore and waste mined, unless an unusual amount of exploratory development is required, and the steel is shortened about ¼ in. at each sharpening. The average length of steel is about 5 ft.; pieces under 2 ft. in length are discarded, so that waste from this source is 40 per cent. Breakage is another cause of loss, and some steel is left behind in old workings, or is lost in the broken ore. The amount of steel in circulation for each drill varies likewise; except in extreme cases, it is from 500 to 1,000 lb., dependent on local conditions.

The leading manufacturers of drill sharpeners are the Ingersoll-Rand, the Gardner-Denver and the Sullivan Machinery companies in the United States, and Holman Brothers in England. The machines all operate on the same general principles as the original Leyner sharpeners, the differences being mainly in details that affect the convenience of the operator. The large machines, such as the No. 50 Ingersoll-Rand and the D.S.-8 Gardner-Denver, have large capacity, and will handle steel up to 2 in. diameter. Intermediate sizes, such as the No. 40 Ingersoll-Rand and the D.S.-3A Gardner-Denver, are designed for steel up to and including 1¼ in. round; and the smaller sizes, such as the No. 34 Ingersoll-Rand and the Class C Sullivan Machinery, are for jackhammer steel. The larger machines sharpen small steel nearly as well as the smaller machines, but are not always as convenient in operation, and they require a larger investment. Recently, Sullivan Machinery has put on the market its Class E sharpener for jackhammer steel, which is suitable for small prospects or road contracts. It weighs only 350 lb., and is comparatively inexpensive.

Drill sharpeners are sold bare; accessories and fittings must be bought to suit individual needs. One dolly is required for each diameter of bit, and it is well to have two sets on hand. A dozen extra dolly springs should also be kept in stock. One set of holding dies is required for each size and shape of steel. One pair of formers for drawing out the wings, and a pair of automatic gaging blocks, are needed. The latter can be depended on for accuracy only when new, but may be used a much longer time if a bit gage, made of ⅛-in. plate and attached to the front of the sharpener, is used in conjunction with them. Gaging blocks, formers, and dies can be built up with the electric welder and then refinished, but one spare of each should be kept on hand. A bit and shank punch, operated by compressed air and mounted on the sharpener, should be provided, and also a set of dies for each size and shape of shank on which lugs or collars must be forged.

Sharpeners should not be mounted directly on a concrete floor or base, but should rest on a cushion made of plank or rubber belting. Otherwise the heavy pounding will crack the base.

Furnaces for forging and tempering are much alike, and in small plants the same furnace may preferably be used for both purposes, but the two operations should not be carried on at the same time, because the temperatures required are markedly different. A good temperature for forging is 1,950 deg. F., and it may be as high as 2,300 deg. F., whereas for tempering it should be around 1,400 deg. F. In both operations a slightly reducing atmosphere should be maintained, so that the steel will not be decarbonized. To create this atmosphere with coke as fuel is impossible, and for this reason coke furnaces have been generally discarded. If natural gas is available, it is an almost ideal fuel, but the fuel most generally used is crude oil or engine distillate. In cold climates 37-38 deg. Bé. engine distillate should be used in winter, but an oil as heavy as 32 deg. Bé. may be used in summer. Consumption varies from 1½ to 2½ gal. per hour, depending on regulation and temperature. The oil is usually fed from a 300- to 500-gal. tank by gravity at a head of 6 ft., or by air pressure at 3 lb. per square inch, and is atomized by compressed air. Burners using compressed air at line pressure are often noisy, and this is undesirable. Low pressure burners are quieter, and, if the blast air is supplied by a blower, the cost is less than for air from the compressor. Reducing valves, such as on gas mains in city service, may be used to reduce the air pressure, and they make possible close regulation. If the mine is on the railroad, to have the oil delivered in 8,000-gal. tank cars and to store it in a 10,000-gal. tank near the track is economical practice. A small pump should be provided, with suitable connections for pumping the oil from the car into the storage tank and from the storage tank to the measuring tank in the shop. Tanks exposed to the sun should be painted with aluminum paint.

The furnace lining should as far as possible be made of standard-size firebrick, 2½ in. by 4½ in. by 9 in. A good quality of brick should be used, because the service is severe, and the more expensive grades are the most economical to use. Bricks should be set in a firebrick cement, such as "Hytempite,"

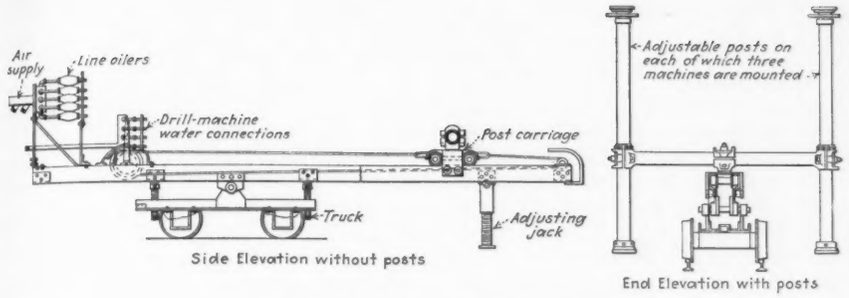


Fig. 60—Montreal mine drill carriage

made by the Quigley Furnace Company, or in some similar material. Ordinary fireclay is not satisfactory. Usually, the furnace is open on one side only; but most designs permit the entry of steel from both sides, if the lining is arranged for it. Although not always provided, a steel hood over the furnace is desirable. An extensible rack for supporting long steel is also a requisite, and is usually included with the furnace.

Close regulation of the furnace temperature is desirable in tempering, and an indicating pyrometer is almost a necessity. The thermocouple is placed at the hot end of the furnace, and the recording mechanism is mounted on the wall, out of harm's way. In addition to the recording mechanism, three electric lamps—one green, one white, and one red—are provided. If the temperature of the furnace is below normal, the green light is lit; if normal, the white; and if above normal, the red. By watching these lights the operator can easily keep the furnace at the proper temperature and prevent overheating. The pyrometer is usually set at 1,450 deg. F.

As an additional precaution each bit should be tested with a magnet before it is plunged in the tempering tank. This practice is based on the fact that, when the temperature of steel is raised to the tempering point, or point of "decalescence," which is about 1,400 deg. F., it loses its magnetism. If the bit has lost its magnetism, its temperature must therefore be between the point of decalescence and the temperature of the furnace; and, if the red light is not lit, it cannot be overheated. By changing from coke to oil furnaces, and by using a pyrometer and magnet for measuring temperatures, one large mine reduced by 45 per cent the number of bits dulled per 1,000 tons of ore produced.

Magnets for testing drill bits can be purchased from the Sullivan Machinery Company, or made in the mine shop. The apparatus consists of a 1-in. horseshoe magnet suspended with points down on small trunnions between two uprights, which have a crossbar on top. Mounted on the crossbar is an incandescent lamp, connected to one wire of the lighting circuit, and on the upper end of the magnet and insulated from it is a flexible strip of copper, attached to the other wire of the lighting circuit. Another strip of copper extends below the lamp in such a position that, if the poles of the magnet are pulled forward, the two copper strips come into contact and light the lamp. A sheet of brass $\frac{1}{16}$ in. thick is mounted on the lower part of the uprights $\frac{1}{16}$ in. in front of the magnet, and the entire unit is set up on a pedestal between the furnace and the tempering tank. When a heated bit not hot enough for tempering is placed against the brass plate, the magnet will be attracted and the lamp will be lit, but if it is hot enough the magnet will not move. This method of testing is particularly useful when steels of different carbon content are used, because the point of decalescence varies with the percentage of carbon.

A method of heating bits for tempering particularly suited to places where unskilled labor must be used is in use on the Rand. The bits are immersed in a salt bath, heated by electricity to 1,470 deg. F., and then plunged in water. The bath is composed of two parts of potassium chloride and one part of barium chloride, and the consumption of salt is small—about one pound per 1,000 bits.

In places where suitable fuel could not be obtained for standard furnaces, muffle furnaces, in which the heated steel does not come in direct contact with the flame, have been tried, but the cost of replacement of the muffles is high, and their use is not extensive.

Pure water is ordinarily used for tempering. The tank in which the bits are cooled should be large enough to prevent much rise in temperature. A screen should be provided a few inches above the bottom, so that the bits will not rest in the sludge that accumulates there. If possible, a small stream of cold water should flow steadily in at one end and out at the other, so that the water will always be fresh and the level will remain unchanged. A screen of perforated plate, $\frac{3}{4}$ in. below the surface at one end of the tank, may be used to stand the hot bits on for their initial chilling, before they are plunged in the tank; or the bit may be chilled by standing it over a rising stream of water. Good results are obtained either way if sufficient care is used to insure uniformity. If the bits are properly heated, however, plunging gives satisfactory results. A tank 18 in. wide, 24 in. high, and 60 in. long, made of $\frac{3}{16}$-in. plate, arc-welded at the joints, is suitable for average conditions. It should have a removable screen or perforated-steel plate, resting on angle irons 6 in. from the bottom, and an overflow 2 in. below the top. Means of support for long drills should be provided.

The best device for tempering drill bits automatically is the Gilman heat-treating machine, manufactured by the Worthington Pump & Machinery Cor-

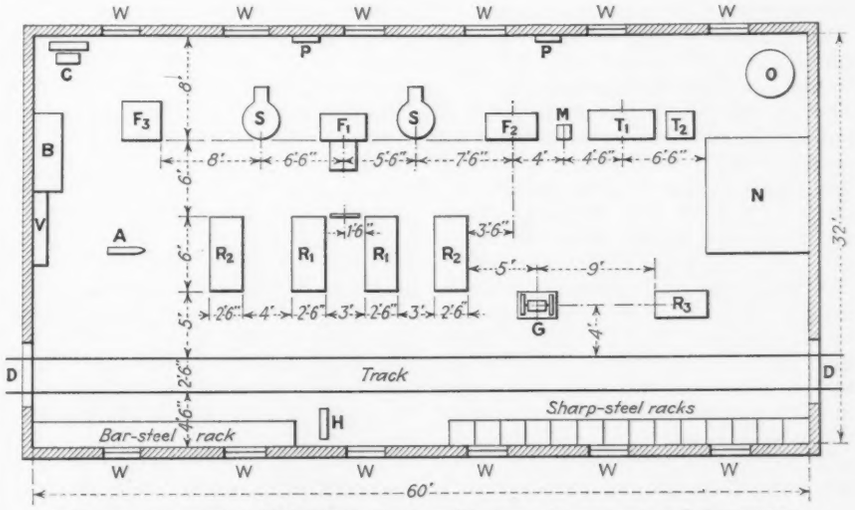


Fig. 61—Drill-steel sharpening shop, with two sharpeners and tempering forge. An automatic tempering machine may be substituted for the tempering forge and tank.

A, Anvil
B, Bench
C, Blower
D, Door
F_1, Sharpening forge
F_2, Tempering forge
F_3, Blacksmith forge
G, Pedestal grinder
H, Hacksaw
M, Magnet
N, Sorting rack
O, Fuel oil storage tank
P, Pyrometer
R_1, Dull-steel rack
R_2, Sharp-steel rack
R_3, Bundling rack
S, Sharpener
T_1, Tempering tank
T_2, Oil-tempering tank
W, Window
V, Tool Cabinet

poration, of Harrison, N. J. This machine is leased on a monthly basis in Mexico, the United States, and Canada, and is sold outright in other parts of the world. The first cost is high, but is fully justified if the number of bits treated daily is large. The machine consists essentially of a heating furnace, a quenching tank, and a number of traveling carriers that pass the steel through the furnace, quench it, and discharge it when cool. The temperature of the furnace is controlled by a pyrometer. Operation of the machine is entirely automatic, the only labor required being that of placing the steel in the carriers and picking it up when finished. The capacity is up to 120 pieces per hour. Ordinarily the furnace is heated by an oil burner, which consumes 2¾ gal. per hour, and requires 300 cu.ft. of free air per minute at a pressure of 8 oz. per square inch; but, as an alternative, it may be heated electrically by "Globars," with a power consumption of 32 to 40 kw. per hour. Maintenance charges on account of breakage of "Globars" are the greatest single item of expense.

An oil bath, containing about 25 gal. of tempering oil, is required for tempering shanks. Fish oil may be used, but special tempering oils are on the market that are better, among which may be mentioned Houghton's No. 2 quenching oil, sold by E. F. Houghton & Company, of Chicago. Oil consumption is low, as the oil may be re-used indefinitely if it is washed periodically.

Other accessories in the shop are a power hacksaw for cutting steel, a pedestal grinder for grinding shanks, and a reamer for reaming out the holes in the shanks. A 300-lb. anvil and 16-lb. sledge and some hot chisels are also needed for straightening bent steel and cutting off spoiled bits. If the shop is large, a separate forge is necessary for this purpose.

Each piece of steel is picked up and put down again a surprisingly large number of times in making the complete circuit from sharpener to drill and back again, and at a mine having even 10 tons of steel in active circulation this is an important item of expense. Much of this handling is unavoidable both in the shop and at the face, but much unnecessary labor is often performed in distribution, and this expense may easily be as great as that of sharpening. If the cage is large, most of the sharpened steel can be laid flat on a truck in the shop as soon as it is tempered and sorted, and can be taken underground and distributed to points near the working places without rehandling. If the cage is small, the steel must be carried upright, and trucks for this purpose are generally used, having four or six compartments, for different lengths of steel. They are difficult to tram, however, on account of high center of gravity. To overcome this difficulty a special design was developed by the United Verde Extension Mining Company, in which the body can be carried in an upright or a horizontal position. A platform truck with hinged or removable posts, such as used for mine timber, is ordinarily all that is necessary; but for centrally located shops, where the steel must be taken to the shafts by motor truck, some form of cradle should be used that can be lifted bodily by chain block or air cylinder and deposited in the body of the motor truck without direct handling of the steel.

For hoisting and lowering steel through long raises, special drill buckets, made of 8- or 10-in. pipe, or light skips with 4- or 6-in. wheels, hoisted by small air hoists, are great savers of labor. In distributing steel underground, common practice is to have all the steel, sorted according to length and section, stored on each level at one or more central points, from which it is distributed to the working places as required. If the working places are numbered, a good plan is to mark every piece of steel with the number of its working place, and to return all sharpened steel to the place where it was used. This is not difficult. Each piece of steel is stamped with its proper number on the shank with numerals ½ in. high. When taken from the tempering tank in the shop, it is placed in the proper pigeonhole of a large rack. The steel is taken from these racks and made up in bundles weighing 100 to 150 lb. apiece, each bundle being held together by a welded ring, 4 in. to 4½ in. diameter and 2 in. broad, a few 1 in. by 1 in. by 6 in. hardwood wedges being used for tightening. In each bundle is always one clean wedge, on which the number of the working place is written with blue lumberman's chalk. No effort is made to keep separate the dull steels from the different working places when they are sent to the shop.

In laying out the shop the equipment should be so arranged that the steel keeps moving in the same direction. The dull steel should be brought into the shop on trucks, sorted according to size and length, and placed horizontally on racks handy to the sharpeners. Work of sharpening will be facilitated if the short steel is sharpened at one machine and the long steel at another. The sharpened steel is placed to cool on a rack near the tempering forge or on a truck that can be trammed to it. After it is tempered, the steel is sorted, bundled, and placed on the proper trucks, or is put in the sharpsteel racks, ready for distribution.

New steel, stored in the shop in moderate quantity, should be cut up in proper lengths with the hacksaw and be shanked and sharpened as opportunity occurs, so that a small supply is always ready for distribution. Bins or racks for new steel, in bars and finished sizes, should be provided.

Whether the drill-sharpening shop should be on surface or underground depends on circumstances. If mining is being done on several levels, the steel is as easily taken to surface as to one central point underground; but if work is nearly all on one level, and if no danger exists of fire or pollution of the mine air, an underground shop may effect a considerable saving. If the shop is on surface, it should be near the most active shaft, to save transportation. If tempering is to be done by eye, it should be so oriented that the tempering department is not on the sunny side.

In planning the equipment for a drill-sharpening shop, some knowledge of the number of pieces of steel to be sharpened daily, and their size, should be acquired and possible future extensions should be foreseen and allowed for. To drive the drill sharpeners at too high speed is a mistake, the quality of the work being more important than the quantity, and the estimates of capacity should be conservative.

The number of bits sharpened per machine varies inversely as the size of the bit. Fully twice as many bits can be sharpened on ⅞-in. steel as on 1¼-in. steel. A safe estimate is 300 to 500 bits per machine per shift, depending on the size of the steel and the skill of the operator. One heating furnace will serve two sharpeners, one tempering furnace will serve three, and an automatic tempering machine will serve four. To do all the sharpening on the day shift is economical, both labor and supervision being more efficient than when two shifts are worked.

To reduce labor required for carrying drill steel to and from the working places in the mine, detachable bits have been tried periodically for several years, but their use has not become general. The Hawksworth detachable bit has been adopted as standard by one large copper company and noteworthy economies are claimed for it. The bits, forged from blanks of cruciform steel, are attached at the end of the drilling shank, provided with slightly beveled morticed joints, formed by milling. (See Chauncey L. Berrien, *Trans.* A.I.M.E. 1930, p. 139.) They are ground to size, and are then hardened and tempered. Dulled bits are reground to ⅛ in. smaller diameter, and are rehardened after grinding. After five reductions in size they are discarded.

The drilling capacity of the detachable bits is about 20 per cent less than that of standard bits, and the cost for steel per cubic foot of excavation is proportionately higher, the saving being entirely in the charges for handling. If no royalty were payable, this type of bit might be more generally adopted; but, if the drill-sharpening shop is well equipped and the work is well organized, any probable saving effected by changing to detachable bits is offset by the royalty. In general, also, detachable bits are not satisfactory in very

hard ground, and in soft ground they are not needed. Their field lies in rocks of medium hardness.

An excellent review of the situation as it is today with regard to detachable bits, by A. H. Hubbell, appears in the November, 1932, issue of *Engineering and Mining Journal*. In this he lists eight different designs and makes of detachable bits, only two of which, however, have been adopted and extensively used by large mining companies. The field for the detachable bit in its present state of development seems to be among small operators and contractors on temporary work, where the first cost of a well-equipped drill-sharpening shop is not justified, or at mines where the cost of distributing steel is abnormally high.

Experiments in the use of hard alloys welded to the cutting and wearing edges of drill bits give promise of success. "Stellite," "Stoodite," and other special alloys have been effectively used for this purpose. They are applied by oxy-acetylene or electric-arc welding, and are finished by grinding with carborundum. Unless properly put on, however, the hard material may crumble under impact, and separate from the steel. The great resistance of the hard alloy to wear reduces loss of gage, and makes possible much greater penetration for each diameter of bit. This has been especially helpful in "deep-hole" drilling, where the depth that can be drilled is limited by the loss in gage. In this work the depth of hole that could be drilled by a bit has been increased six to eight times through the use of "Stellite."

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Doubled Azimuth Traverse

THE METHOD of doubled deflection angles has the advantage over the plain azimuth traverse of approximately doubling the accuracy of the transit, and of taking care of maladjustment of collimation, according to M. B. Evans, of San Jose, Costa Rica. Against it, however, as usually applied, are the initial error at each set-up of setting the vernier and the final error of reading it, unless a continuous vernier is carried. This practice will also tend to eliminate another source of error—namely, the confusing of small "lefts" and "rights," which is one of the important defects of the method. At the end of the day, a mass of calculations have to be made to obtain the bearings of the courses, with nothing in the notes to serve as a quick and continuous check on the calculations.

If, however, at the start of a traverse, the vernier is set on the doubled azimuth of the backsight, less 360 deg. when in the third or fourth quadrants, and a continuous vernier used as for doubled deflection angles, then at each set-up the vernier, plus 360 deg. when necessary, will be at the doubled azimuth of the course.

At each set-up the two readings of the vernier, in conjunction with the final reading of the previous set-up, give the two values of the deflection angle; and, using their average in the calculations, the calculated bearing of every course must check with the half, plus 180 deg. when necessary, of its corresponding reading of the vernier. Customary calculations by deflection angles might therefore be said to be a check on the calculation of the azimuths, as determined by the direct method of halving the vernier.

In setting a station on a branching line, the branching station should be located first, and then used as a backsight for locating the next station on the main traverse. In taking a set of side shots, the succeeding main station should be taken first and a note made that the side shots are based on that reading of the vernier. Except for faulty handling of the instrument, the vernier should give its original reading when the telescope is finally resighted on the succeeding station before moving ahead.

Advantages of the foregoing method, as compared to the method of doubled deflection angles are: (1) an elimination of the error of setting the vernier; (2) practically an elimination of the error of reading the vernier, because a half minute dropped on one reading may be recovered on the next, resetting in a less serious probability of error than that of the "compensating error of reading" of the method of doubled deflection angles; (3) no excuse exists for confusing "rights" and "lefts"; (4) and, most important, the calculated bearing of each course is simply and instantaneously checked. In short, with the same work, the method results in increased accuracy of angular measurement and an absolute check on angular calculations.

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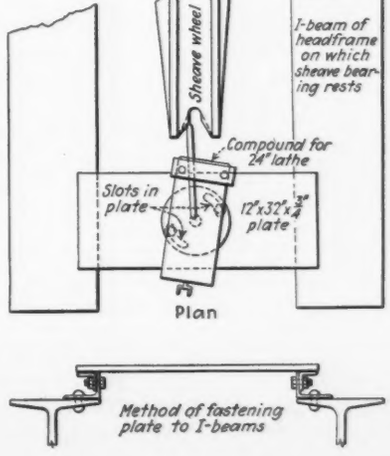
Turning Sheave-Wheel Grooves

A METHOD used by Miami Copper for turning and rectifying the cable grooves of large sheave wheels is described in the Nov. 9, 1931, issue of *Engineering and Mining Journal*, but the frame described in the article is now seldom used, as a method has been developed whereby the job may be done with the sheave wheel in place. A plate is fastened across the headframe I-beams that support the sheave bearings, and a 24-in. lathe compound is bolted to the plate. The sheave wheel is moved by the hoist while the groove is being turned, and can be moved in only one direction—toward the hoist; and when, during the cutting, the skip has reached the top of its run, it must be dropped back to the bottom of the shaft before cutting is resumed. The cutting tool, made of $\frac{3}{4}$x$1\frac{1}{2}$-in. self-hardening steel, is held in place by straps and $\frac{3}{4}$-in. studs, which give greater stability than would a tool post. The cutting edge is wide, so that the groove in the sheave wheel will be smooth. One end of the tool has the cutting edge on the right-hand side, to turn the right side of the groove; and the other end has the cutting edge on the left side, for the left side of the groove. Otherwise, two cutting tools would be necessary—one left-hand and one right-hand. The bolt holes in the steel are slotted to allow adjustment of the direction of feed of the compound. The ore-hoist ropes are $1\frac{3}{4}$-in. diameter, and the grooves are made $1\frac{7}{8}$-in. diameter.



Centrifugal Concentration of Placer Gravel

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THE REVIVAL of placer mining throughout the West has demonstrated that the bonanza dirt found by the Forty-Niner is a thing of the past. Thousands of men, and not a few women, in an attempt to provide a living for themselves, have turned to the gold pan. Some have obtained a meagre living thus; others have quit in despair.

Placer mining today is a material-moving problem. No longer can one follow the rich run of gravels and extract the gold usually associated with the memories of the olden days. Modern materials-moving machinery, in conjunction with a simple gold concentrator such as is described in this article, now makes possible the handling of the large yardage necessary to work the low-grade placer deposits profitably at a cost of a few cents a cubic yard, with a minimum water consumption. With a plant designed to work in conjunction with a 2½-yd. dragline, the material can be dug, passed through the concentrator, and, by means of a stacker, put back into the pit from which it was dug, with a minimum amount of work.

Many placers in the West are too low in grade to be exploited on a small scale, and too far from water to sluice or hydraulic. These deposits, if properly equipped with a portable plant, will yield a good profit on the investment. This investment is in no manner comparable to the investment required to bring the low-grade porphyry coppers into production. A capital investment of less than $50,000 will equip a property to treat 5,000 yards a day. This, perhaps, would not cover the full cost of an expensive water development, but water development is of small consequence if the yardage is sufficiently large to justify operation.

More placer-mining ventures have failed because of faulty sampling than from any other cause. Sampling is usually done by drilling, or by rocker and pan, neither of which method handles a volume comparable with the deposit being investigated. The sampling operation should, in general, follow the same process as is proposed for the commercial operation.

This article describes a plant, shown in an accompanying illustration, set up at Oro Blanco, Ariz., about 1½ miles north of the Mexican boundry, for the purposes of sampling widely separated placer deposits. As the ground was not in one block, a stationary plant, to which the material could be hauled by truck, was considered less expensive than a portable plant; because the water, a determining factor, could be supplied to the stationary plant with only a few hundred feet of pipe; whereas, with a portable plant, many thousand feet of pipe had to be laid at a cost greatly exceeding the cost of hauling the material by truck during the sampling period.

The gravels to be sampled are loaded into trucks by a ⅜-yd. power shovel of the Fordson type. This unit in normal digging can deliver 300 cu.yd. per day; in heavy digging, as was encountered in some parts of the property, the shovel was unable to maintain this production without preliminary loosening of the gravel with powder. A larger shovel could dig the material easily. The low cost of operation of such unit, plus its great mobility, make is particularly adaptable for handling only 300 cu.yd. per shift. For the extraction of the gold from the gravel, a centrifugal concentrator developed by me was used. This consists of a drum *A*, as shown in Fig. 1. The placer material is fed into the smaller inlet opening *E;* and the ore, divested of metallic content, passes out the larger or discharge opening.

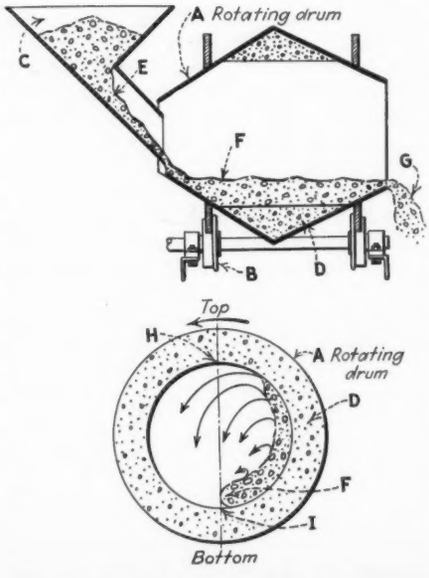


Fig. 1—Diagram of operation of centrifugal concentrator

When a bucket of water is swung in a vertical plane, the water will be retained in the bucket if the speed of rotation is rapid; and, when this speed is reduced, the water falls. At a certain speed the water will be just ready to fall. The drum is rotated so that this condition obtains inside it, producing what we call a zone of unstable equilibrium *F*, which zone is held by speed control to a diameter slightly larger than the outlet of the drum.

The screened ore containing free gold is fed into the small end of the drum. As soon as the free metallic particles come under the influence of centrifugal force they are drawn outward, penetrating the soft, spongy zone, and find lodgment in the collecting cushion *D* outside this zone, which is completely under the influence of centrifugal force, where they are embedded and continue to penetrate deeper and deeper into the absorbing cushion *D* as the diameter of their path of revolution increases. The ore thus freed of its metallic content passes out of the drum. Experience has demonstrated that a large percentage of the deposition takes place immediately inside the entry end of the drum.

To clean up, usually once a day, the machine is stopped, and the material with its embedded gold is removed. In the Oro Blanco plant, which has a drum 4 ft. long, the concentrate amounts to about 8 cu.ft. It is further reduced to about ½ cu.ft. by a smaller concentrator operating on the same principle. From this concentrate all the visible gold is removed by panning; the microscopic gold, by melting.

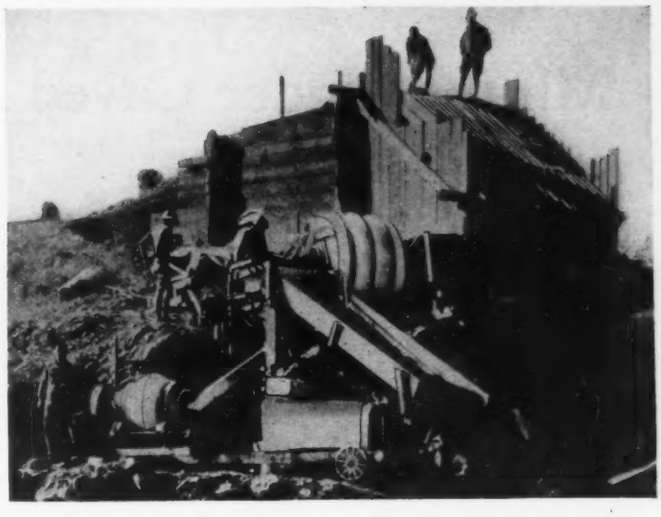
Before the gravel enters the concentrator it is given primary treatment, consisting of the usual mill practice of grizzlying off large boulders, scrubbing to break down clay, and screening to minus ⅜ in. These operations are standard practice, but the scrubber is built on the principle of a concrete mixer. The gravel is not only churned in the drum but buckets lift the material to the top and drop it on the bottom, giving a much more vigorous grinding and scrubbing action. Buckets also lift the material near the discharge and pour it on a chute in the same manner as occurs in the discharge of a concrete mixer; this saves head room—generally an important consideration. At this plant the material leaves the scrubber at a higher elevation than it enters. The discharge from the scrubber is on to a vibrating screen; the

undersize then goes to the concentrator, which discharges into a sluice box. Comparison of gold caught in the sluice compared with the gold recovered by the concentrator indicates the efficiency of the concentrator.

When the plant was delivered to the mine owners a special acceptance test was run that showed a saving of 97 per cent. This high recovery was due, no doubt, to the thorough manner in which the placer dirt was scrubbed and the gold liberated. I believe that the extraction obtainable depends to a large extent on the liberation of gold from the containing material.

Water consumption of the concentrator is low. About 40 per cent of the pit run is rejected as oversize gravel, which shows that the concentrator treats about 180 cu.yd. during the normal run per shift. Water consumption is 25 gal. per minute or 66 gal. per yard treated—about one part of water to three parts of placer dirt, which, in this case, contains much clay.

Cost of milling and concentrating at Oro Blanco is lower than at any other plant handling comparable yardages in the West. The primary treatment, feeder, scrubber, screen, and complementary apparatus, all driven from one gasoline engine, require about 10 hp. The concentrator, with a separate engine, requires 5 hp. Gasoline in this isolated region costs 20c. per gallon, and the consumption is 20 gal. per shift. Oil costs $1, and the operator is paid $5. Because of low capital investment, the daily interest charge is less than $2. The plant has not been in operation long enough to give precise data on repairs, but this cost should not exceed normal replacement owing to wear on similar installations. Total milling cost is not more than $15 per shift, or 5c. per cubic yard. This same type of installation, with only added increment costs, could handle 1,000 yd. per shift, with a milling and concentration cost of 2c. per yard.



Oro Blanco placer sampling plant

A sampling operation such as this, which handles large yardages at a low cost, possesses considerable advantages over other methods of sampling. In the first place, the amount of material handled is large enough to give a fair estimate of the value of the ground; secondly, the gold recovered is a sizable amount, repaying a portion of the sampling costs. With low digging costs such as characterize this operation, together with low milling and concentrating costs, the ground is sampled at a profit, instead of a loss, as is usually the case.

This Oro Blanco sampling plant, in one 8-hour shift, will test 300 cu.yd. of placer dirt, the volumetric equivalent of more than 40,000 linear feet of 6-in. drill holes. A 30-day run will test 9,000 cu.yd., nearly 1 per cent of a deposit of 1,000,000 cu.yd. The material is handled with a power shovel, which reduces the possibility of a small pocket of gold raising the average out of proportion to the quantity. Such an operation will give results that can be used as operating data, and not merely as assumption, as is usual with sampling on which a large operation may be based.

Metal-Mine Standardization Projects

SEVEN metal-mine standardization projects, initiated or completed, are included in a list recently issued by the American Standards Association. One of these covers methods for screen-testing of ores, approved by the Association in 1932. In so doing, attention was called to the work of the Sectional Committee on Sieves for Testing Purposes, which, it is said, may ultimately affect some of the provisions of this standard. The approval, however, does not prejudice future recommendations as to standard practice that may be received from the committee.

A standard in regard to wire rope for mines was approved in 1927 and has had wide distribution. Another, on the construction and maintenance of ladders and stairs for mines, was approved tentatively in 1928. Its provisions had been correlated with those contained in the Safety Code for the construction, care, and use of ladders. Should the latter be revised (and this is now under consideration), the question of revising the mine-ladder code might arise. A standard, approved in 1930, covers fire-fighting equipment in metal mines. It is said to have been favorably received by the industry, but its general adoption has been hindered by prevailing conditions.

Underground transportation in metal mines is covered by a standard, approved in 1928. Revision has been under consideration for some time. Plans for expanding the project have been proposed, but little progress has been made. Mechanical loading underground in metal mines is covered by a standard approved in 1928. It was intended to indicate the general lines that standardization might follow to be more effective in such work. Revision and extension are being considered.

The mine-timbering standardization project was initiated in 1930. A draft covering the preservative treatment of mine timbers was circulated the following year. The sectional committee concerned is being reconstituted. The standard most recently approved by the Association is that covering safety rules for installing and using electrical equipment in metal mines. It received approval as American accepted practice in August, 1932.

Repairing Doors of Air-Dump Cars

WHEN an 8-yd. dipper loaded with 14 tons of iron ore is accidentally dropped on the door of a 30-yd. air-dump car a bad dent results. These doors, more than 30 ft. long, have a 6-in. channel at the top and ¼-in. plate sides, which are well braced. To rebuild the door by cutting it apart and straightening the members costs about $260. A scheme to do the job for $60 and save the $200 was observed at one of the Mesabi mine shops. A rectangular section of the door inclosing the dent was cut out with a gas cutting torch. Then a new section was made up of new parts and electrically wedded into the place. The repair makes the door as strong as new, leaves no patchy appearance, and new sections can be welded in as needed.

Mining Under Caved Areas

F. S. McNicholas

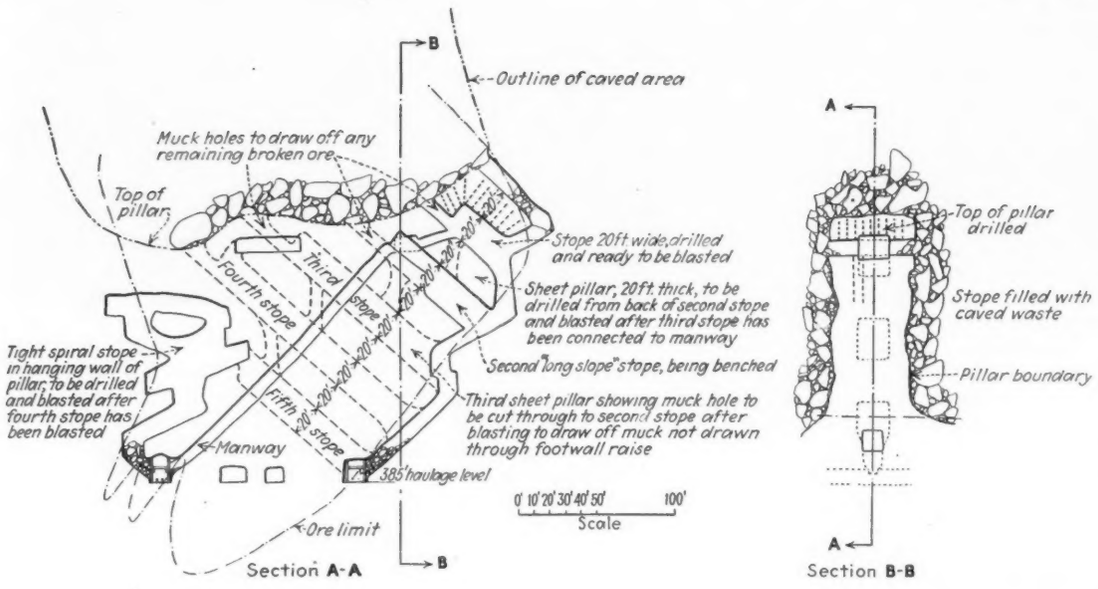

B
Outline of caved area
Muck holes to draw off any remaining broken ore
Top of pillar
Fourth stope
Third stope
Stope 20ft wide, drilled and ready to be blasted
Sheet pillar, 20ft thick, to be drilled from back of second stope and blasted after third stope has been connected to manway
Second "long slope" stope, being benched
Third sheet pillar showing muck hole to be cut through to second stope after blasting to draw off muck not drawn through footwall raise
Tight spiral stope in hanging wall of pillar, to be drilled and blasted after fourth stope has been blasted
Fifth stope
Manway
385' haulage level
Ore limit
0' 10' 20' 30' 40' 50' 100'
Scale
Section A-A
A
Top of pillar drilled
Stope filled with caved waste
Pillar boundary
Section B-B

Fig. 1—Illustrating method of mining pillars surrounded by caved waste

A METHOD of mining pillars surrounded by caved waste, and stope bottoms covered by waste, has been developed at the Hidden Creek mine of the Granby Consolidated Mining, Smelting & Power Company, at Anyox, B. C. It is described in the following. The reader should note that pillars left in large orebodies are from 40 to 60 ft. thick and permit rather large openings to be made in them.

For mining vertical pillars surrounded by caved waste (Fig. 1) a 7x12-ft. muck raise is driven in the pillar and along the footwall of the ore, to the top of the pillar. Near the top of the raise an inclined "long slope" stope[1], about 20 ft. wide, is driven in the pillar parallel to it and toward the hanging wall. This stope makes contact with a hanging-wall or other manway raise and continues to the top of the pillar, but does not break through into the surrounding caved waste. The stope is then either slabbed down from the back, or benched once to give a stope 20 ft. high.

The next stope is then started 30 ft. below this and parallel to it, and is driven in the same manner. After contact is made with the manway, the back of the upper stope is drilled with 20-ft. holes which are blasted electrically and all at the same time and the resulting broken ore is drawn. The lower stope serves as a manway to the draw point so that the broken material may be bulldozed, plugged, and drawn. When the muck begins to run low in grade, large boulders are allowed to remain in the neck of the raise above the draw point and effectively cut off further flow of waste. At the top of the second stope muck holes are driven to draw any ore that has not been drawn through the footwall raise.

The same process is repeated until the entire pillar is mined.

In mining bottoms under old stopes filled with caved waste (Fig. 2) the procedure is similar: either a footwall raise or a tight spiral raise is driven through the area to be mined. If a footwall raise is used, manways to the draw points must be supplied. If a tight spiral raise can be used, no additional manway is needed, because broken material will draw down the inside of the spiral, leaving a manway around the periphery of the spiral.

Near the top of the area to be mined a stope is cut out, about 20 ft. high, leaving a 25-ft. back between the stope and the caved waste. A stope is then cut out about 30 ft. below the first one, the original raise being left unenlarged between the stopes. This gives a "bottle neck" below each stope, which may be choked almost at will to cut off the flow of waste. The under stope furnishes a safe entry and working place for bulldozing and plugging of broken material.

Then the upper stope is drilled with 20-ft. holes and blasted in one simultaneous blast electrically. Broken material is drawn until it runs low in grade, when it is choked by allowing big boulders to remain in the bottle neck of the raise. In the second stope and the subsequent stopes, muck holes are broken through to the caved material and drawn until they run waste before the stope is blasted. Similar procedure continues from the top downward until the entire area is mined.

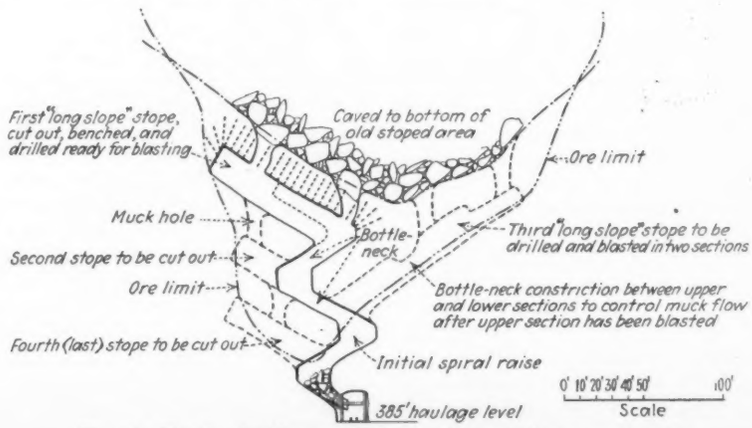

First "long slope" stope, cut out, benched, and drilled ready for blasting
Caved to bottom of old stoped area
Ore limit
Muck hole
Bottle-neck
Third "long slope" stope to be drilled and blasted in two sections
Second stope to be cut out
Ore limit
Bottle-neck constriction between upper and lower sections to control muck flow after upper section has been blasted
Fourth (last) stope to be cut out
Initial spiral raise
385' haulage level
0' 10' 20' 30' 40' 50' 100'
Scale

Fig. 2—Sketch showing method of mining bottoms under old stops filled with caved waste

[1]See "Long Slope Mining at Anyox," by F. S. McNicholas, *E.&M.J.*, November, 1932, pages 567-568.

Employee Housing—VI

A. H. Hubbell
Associate Editor

A residential street in Arvida, Quebec

IN THIS, the concluding installment of this series of articles on employee housing, are presented data and illustrations on housing in two noteworthy communities—one mining and the other metallurgical in the character of their respective industries—at Boliden, Sweden, and Arvida, Quebec. Both are in northern climates. The article concludes with illustrations of the housing at a well-known nitrate plant in Chile.

Housing of Aluminum Company of Canada, Ltd., at Arvida, Quebec, Canada. In August, 1925, Aluminum Company of Canada, Ltd., purchased approximately 6,500 acres of land on the south bank of the Saguenay River about

Exterior view of a house shown in plan in Fig. 57. Houses having this floor plan may differ in elevation. See text.

35 miles east of Lake St. John, and in latitude approximately 48 deg. 30 min. N., as a site on which to construct a large aluminum reduction plant and a town for its employees. The land purchased is irregular in shape and is approximately 4 miles long in its greatest east-west dimension and 3¼ miles north and south. For the most part it is a fairly level plateau, about 300 ft. above the river, which is cut by numerous deep ravines extending across the property in a north-south direction. This area is thought sufficient for a city of 50,000 people.

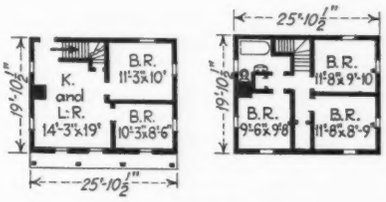


Fig. 57—Floor plans of two-story house, having six rooms and bath, for workmen. Aluminum Company of Canada, Arvida, Quebec.

Following a topographic survey a general plan was prepared for the entire area, showing the principal arteries of transportation, streets, building lots, and the location of the principal buildings such as schools, churches, and other community structures. To date only a relatively small part of the area has been actually built up. As the plan is extended, additional sections of the town will be built following the general plan. Fig. 56 shows the town and plant buildings as they now exist. The average building lot within the residential areas is 50x110 ft. In the business section the lots are 25x100 ft.

The climate at Arvida is very severe. Snowfall is heavy in the winter and rainfall is heavy during the remainder of the year, especially in May, June, and October. Precipitation averaged 38.4 in. in 1930 and 33.83 in 1931. Winter conditions prevail from Nov. 15 to April 15. The average temperature for January, 1931, was 3 deg. F. Minimum temperatures are as low as minus 40 deg. F. The warmest months are July and August. The average temperature for July, 1931, was 65 deg. F.

Bunk houses sufficient to provide accommodation for 800 men were built in 1925 for use during the construction period. They have not been used, however, since the heavy construction work was finished in 1928.

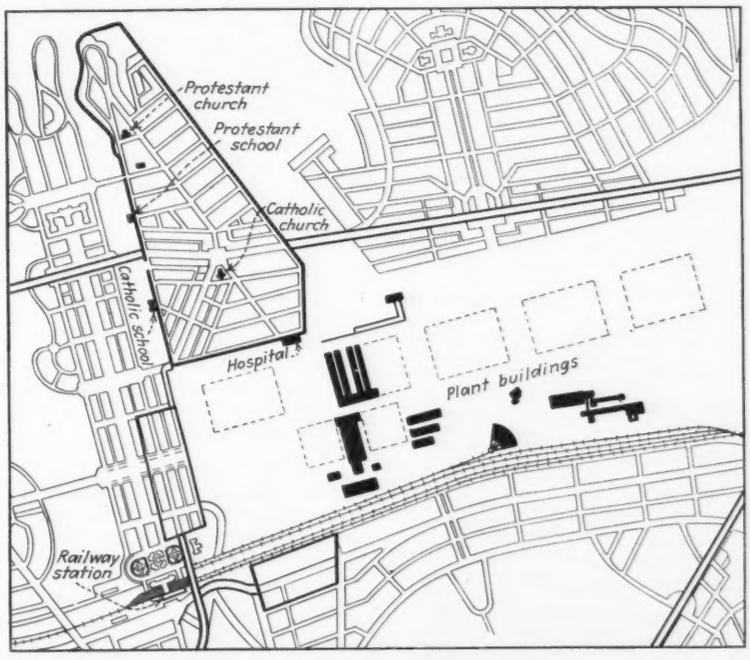


Fig. 56—Arvida, Quebec. Central part of original tentative layout. The heavy lines enclose the residential areas thus far developed. Plant buildings erected, in solid black. Churches, schools, hospital, and railway station are shown.

Previous installments have dealt with employee housing in Northern Rhodesia; Newgulf, Tex.; Matahambre, Cuba; Achotla and Monterrey, Mexico; Pecos Valley, N. M.; Trona, Calif.; Tyrone, N. M.; Clarkdale, Ariz.; Northern Manitoba; Mount Isa, Queensland, Australia; Chuquicamata, Chile; Copperton and Bingham Canyon, Utah; Ruth, Nev.; Conda, Idaho; and Langeloth, Pa.

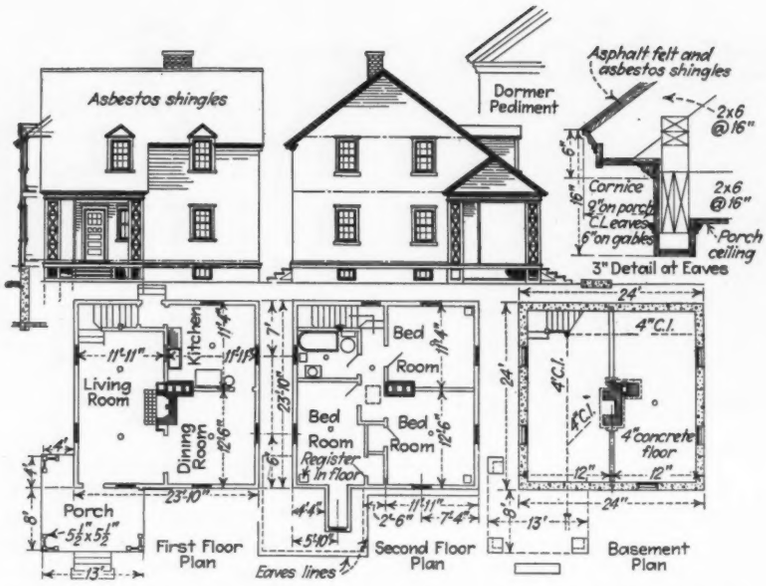


Fig. 58—Two-story house for workmen, in floor plan, elevations, and details

One of the workmen's houses having the floor plans shown in Fig. 58.

In 1926, 282 houses were constructed. Of this number 101 resemble in a general way the type of house commonly seen in the small towns and farming districts throughout Quebec. These houses have the curving eaves and the large combination living room, dining room and kitchen characteristic of such Quebec houses. A large cook stove situated in this room is usually sufficient to heat the entire house. This type of house is illustrated in Fig. 57. It was adopted by the company as being best suited to the French-Canadian labor the company expected to employ and so far has proved more popular than any other with those employed in the plant.

Figs. 57 to 63 show seven houses of different types as distinguished by size and floor plans. Various elevations were designed for each floor plan, so that from the seven types illustrated 27 subtypes have been developed. The houses illustrated in Figs. 57 to 61 are occupied by workmen, foremen, higher-paid mechanics, and some of the clerical staff. Those occupied by the latter have a somewhat better interior finish. Those illustrated in Figs. 62 and 63 are occupied by the principal executives, members of the technical staff, and members of the office staff.

Exterior view of house shown in Fig. 61.

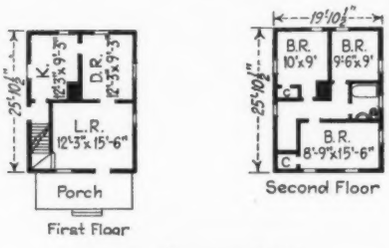


Fig. 61—Six-room house, Arvida. The so-called "F" type. Floor plans.

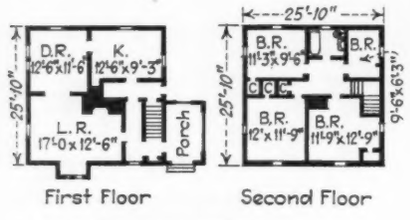


Fig. 59—Another type of workman's house, Arvida. Floor plans.

House for workmen, Arvida. Floor plans shown in Fig. 60.

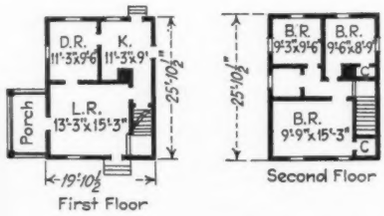


Fig. 60—Floor plans of another group of workmen's houses that may differ in elevation. This house and those shown in Figs. 57-59 are all of the same construction.

House shown in floor plan in Fig. 59. For workmen.

Owing to the number of different elevations, the housing exhibits a pleasing variety of design. All of the houses are of frame construction with concrete basements under the entire house. Studding and sheathing are of B. C. fir. The siding is beveled and practically all of Western cedar. Every roof is fireproof, either of rigid asbestos or aluminum shingles, colored gray, green, red, plum, brown, and black. British Columbia fir flooring was used in all houses except those illustrated in Figs. 62 and 63, in which they are of birch. All floors are double with building paper in between. Interior walls are all plaster on plasterboard. Floors are all stained and walls painted.

The houses illustrated by Fig. 57 are heated by stoves; those in Figs. 58 to 61 have pipeless hot-air furnaces, and those

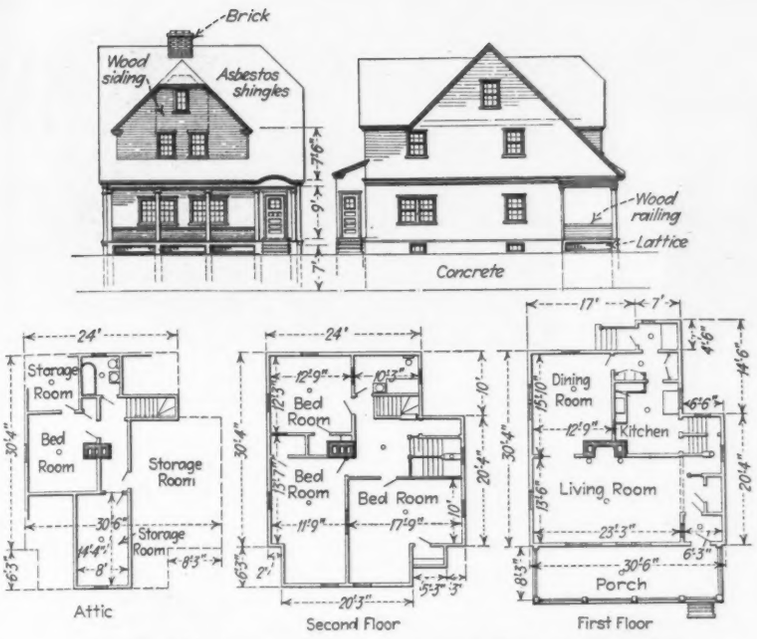


Fig. 62—This type of house and the one shown in floor plan in Fig. 63 are occupied by principal executives and technical and office staffs at Arvida.

plans and the building of the houses were done by the company's field organization at Arvida, the office work being in charge of J. W. Schreiber, chief engineer at Pittsburgh, and the field work under H. R. Wake, superintendent of properties at Arvida.

The cost[1] of workmen's houses ranges from $4,700 to $5,800, as estimated in 1926. The lower of these figures, which here is equivalent to 37c. per cubic foot, pertains to the house shown in Fig. 57, the higher, equivalent here to 31c. per cubic foot, to the one in Fig. 59. The dwellings in Figs. 58 and 60 cost $5,200 (31c. per cubic foot) and $4,800 (34c. per cubic foot), respectively.

The so called "F" type, shown in Fig. 61 and in a photographic reproduction, cost $5,200, or 42c. per cubic foot. The house shown in Fig. 62, of which there are only two, cost $14,000, or 48c. per cubic foot. That seen in floor plan in Fig. 63 cost $10,000, or 49c. per cubic foot.

Housing at Boliden, Northern Sweden. The manner in which a

in Figs. 62 and 63 are heated by hot water.

In this climate insulation against the loss of heat is necessary. The houses in Figs. 57 to 61 have either fiber-board or shavings over the second-floor ceilings to prevent the escape of heat. These houses do not have insulation in the walls. The walls of the houses in Figs. 62 and 63 are insulated with precast slabs of Insulex placed between the studding. A ¾-in. air space was left on each side, and the joints between the slabs and studding are filled with Insulex mortar. Over the second floor ceilings 2½-in. of dry Insulex was used.

Each house is supplied with running water and electricity. Kitchens are equipped with enameled sink, drainboard, and hot-water tank. Bathrooms are equipped with lavatory, bath, and toilet. Many tenants use electric stoves for cooking.

The present population of Arvida numbers a little over 1,700 persons. There are 284 families. Only white labor is employed, and most of the employees are married. Most of the single men are cared for in private families or boarding houses.

To the late Hjalmer E. Skougor, consulting architect, is to be credited the basic planning. The detailing of the

[1]In presenting the costs of the housing at Copperton, Utah, Conda, Idaho, and Ruth, Nev., in the preceding installment of this article, which appeared in the February issue, the respective years for which the costs were calculated were not stated. The eight-room bunk houses at Copperton were built in 1929. The four- and five- room cottages were erected in the years 1926 to 1930, and the costs given for the latter were the estimated average building costs for those years. Bunk houses at Ruth were erected in 1924 and the cottages in 1929. The houses at Conda were built at various times from 1920 to 1928 and the year 1927 may be taken as an average.

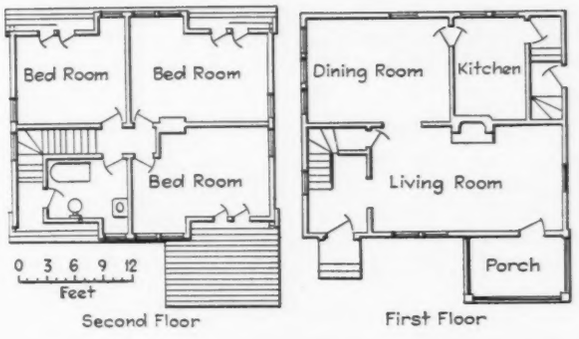


Fig. 63 — Floor plans of a type of house also allotted to executives and technical men. See Fig. 62.

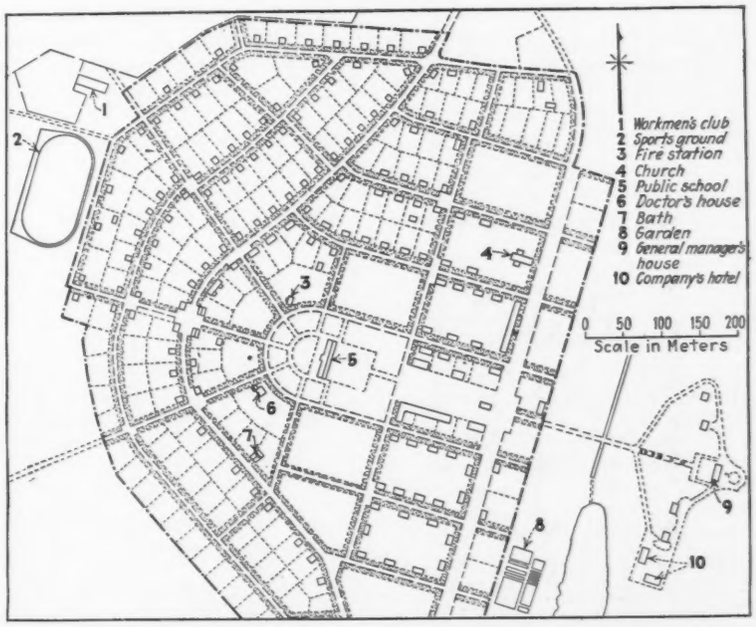


Fig. 64—Layout of Boliden town, northern Sweden. 1, Workmen's club; 2, Sports ground; 3, Fire station; 4, Church; 5, Public school; 6, Doctor's house; 7, Bath; 8, Garden; 9, General manager's house; 10, Company hotel.

General manager's residence, Boliden mine.

Typical staff dwellings, at the Boliden mine.

modern community has been established at Boliden, in northern Sweden, in what was a wilderness when the deposits now worked by the Boliden Mining Company were discovered in 1924, was described in the July, 1932, issue of this magazine by E. Wesslau, superintendent of the mine. Of what was then written it is worth repeating that no building was permitted on the company's land until a complete townsite plan with adequate regulations had been prepared. The plan in shown in Fig. 64. Streets and roads were than built by the company, and a water system, including a chlorination and filtering plant sufficient for the needs of a community of 2,000 people, was installed. Of the houses provided, 38 were built for foremen and workmen. These contain 87 large apartments, most of which have two rooms and a kitchen. In addition, 52 larger single rooms were provided for unmarried workmen. For the technical and operating staff, 17 houses of varying size were built; likewise a hotel and a mess house. Some of these dwellings are shown in cuts which have been made from photographs supplied through the courtesy of the American-Swedish News Exchange, New York.

One family dwelling, Maria Elena plant. Floor plan shown in Fig. 65.

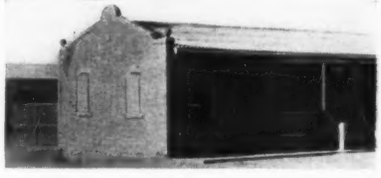
The climate in this region is mild, considering the latitude of approximately 65 deg. N. Temperature in the four summer months, as recorded, has averaged 52 deg. F., the maximum being 80.6 deg. F. During the four winter months the recorded average has been 48.4 deg. F. and the minimum minus 18.4 deg. F. Information on precipitation is not available.

The houses are of wood-frame construction with concrete foundations. Siding is of boards and planks and the inside walls are of boards. The roof is of "Eternite" tile.

To afford additional protection against heat and cold, the floor is double, with peat mold between. Between the walls various kinds of straw mats are used.

The water supply is filtered and chlorinated, and piped from a centrally located tank. Heat is provided from a central source. Toilet and bathing facilities include water closets, "bathing" rooms, and a public bath house.

Housing of Anglo-Chilean Consolidated Nitrate Corporation, Maria Elena Plant, Chile. In an earlier installment of this series of articles (*E.&M.J.*, October, 1932) the housing of the Chile Exploration Company at Chuquicamata, Chile, was described.[2] In contrast to the site at Chuquicamata, which is in the Andes 9,000 ft. above the sea, that of the Maria Elena nitrate plant is situated at approxi-



One of the attached dwellings for married staff members. Plan shown in Fig. 66.

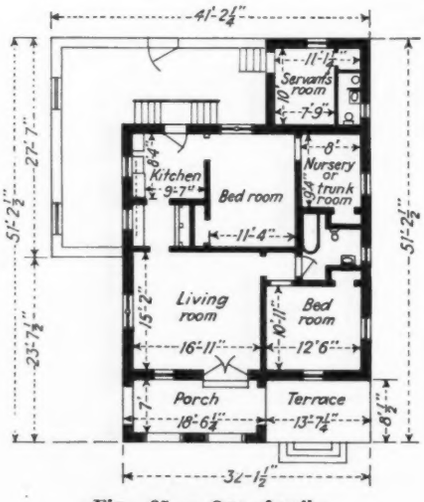


Fig. 65 — One-family dwelling for married members of staff. Floor plan. Anglo-Chilean Consolidated Nitrate Corporation, Maria Elena plant.

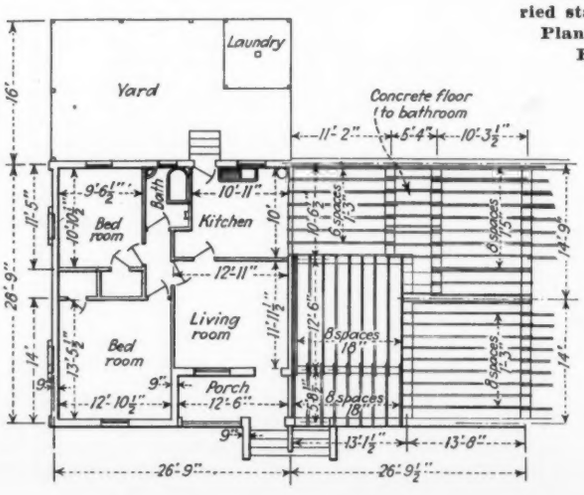
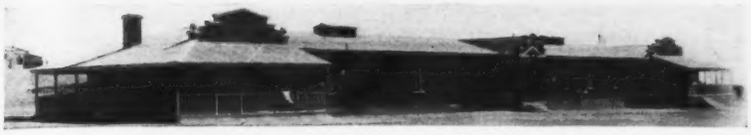


Fig. 66—Two dwellings (plans) of block of six, for married members of staff. Maria Elena plant, Chile.



Eighteen-room staff house. See also Fig. 67.

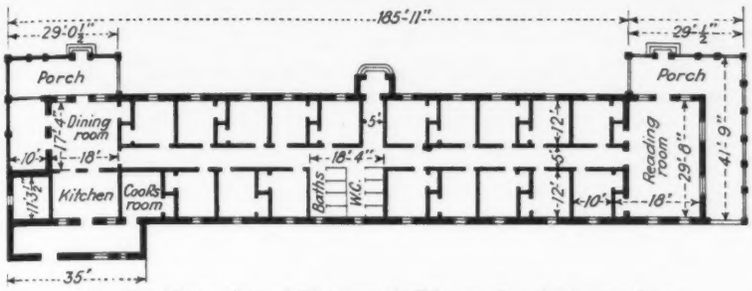


Fig. 67—Floor plan of 18-room staff house for single members of staff. Maria Elena plant.

mately 3,600 ft. The task of planning the community at the latter place, as well as that of designing the housing, was also done by the late Hjalmer E. Skougor, whose work at Arvida, Quebec, is mentioned elsewhere in this article. The illustrations here shown, Figs. 65-71 and the accompanying halftones, were supplied by Mr. Skougor, who had also planned to give me adequate descriptive and cost data to accompany them. Because of Mr. Skougor's death and for the reason that the affairs of the Anglo-Chilean company have since been ordered liquidated by the Chilean Government, making the desired information difficult to obtain from official source, Mr. Louis Ware has kindly supplied the facts here presented. Doubtless the community depicted will be occupied again when the demand for nitrate revives.

The climate at Maria Elena is dry, precipitation being almost zero. Rain is rare and snow unknown. The temperature in summer rarely goes above 100 deg. F. In winter it occasionally falls below the freezing point.

The dwellings illustrated have foundations of stone and concrete. Walls are usually of concrete, the inside being whitewashed in the houses built for workmen and plastered in those allotted to the staff. Many of the workmen's houses have sidewalls of 2x4-in. studding with wire netting between and a lean mixture of concrete on either side of the wire. Such lumber as is used is principally Oregon pine. In some buildings the walls are of adobe with a stucco finish. Roofing of corrugated iron, commonly covered with a layer of adobe as heat insulation, is used on the workmen's houses. Staff houses are roofed with "Italite," a corrugated tile-like sheet imported from Italy.

Both salt water and distilled water are piped to the houses, which are wired for electric light. Staff dwellings have private toilets and bathrooms. Workmen have community facilities.

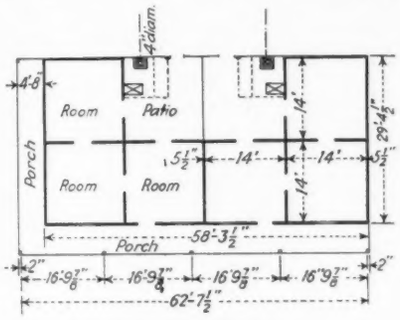
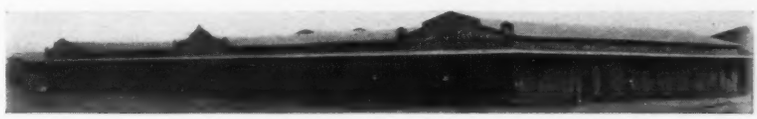


Fig. 68—Two three-room attached dwellings for married laborers. Maria Elena plant.



Living quarters, single mechanics. See Fig. 70.

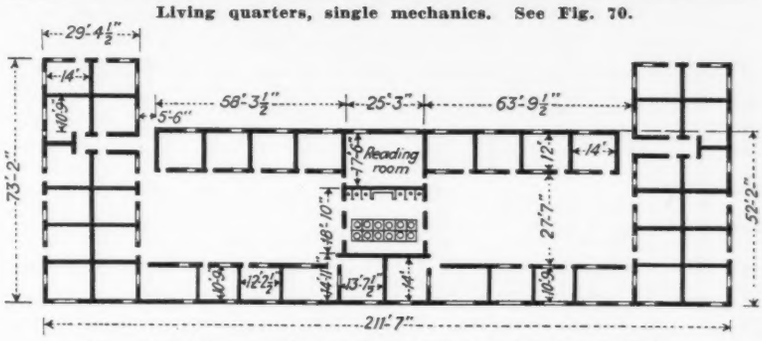


Fig. 70—Living quarters, floor plan, for single mechanics. Maria Elena plant.

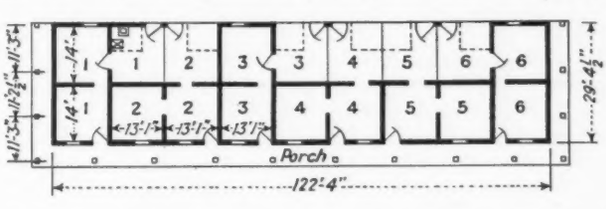


Fig. 69 — Block of six two-room living quarters for married laborers. Maria Elena plant.

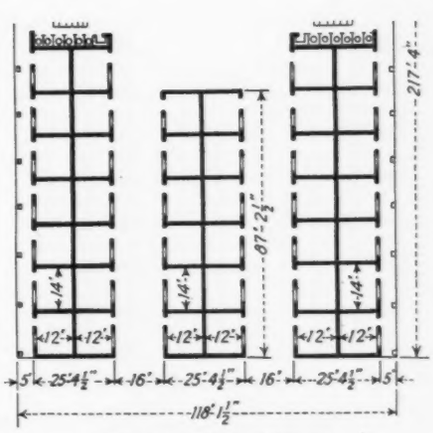


Fig. 71—Floor plan of one half of a unit of living quarters provided for single laborers, Maria Elena plant.

Living quarters, single laborers. See Fig. 71.

[2]With reference to the text of the article on the housing of the Chile Exploration Company, at Chuquicamata, which was published in the October issue, W. D. B. Motter, Jr. assistant to the vice-president of Chile Exploration, has commented as follows: "Referring to Fig. 40 (block of three-room houses built for married workmen), page 520, you say 'These have inside walls lined with sheet steel, but lack electric light, toilet, central water and bath.' While it is true that the individual apartments do not have these facilities, the blocks are arranged so that they are available to groups of apartments."

Genesis of the Diamond

A review by Dr. F. L. Ransome

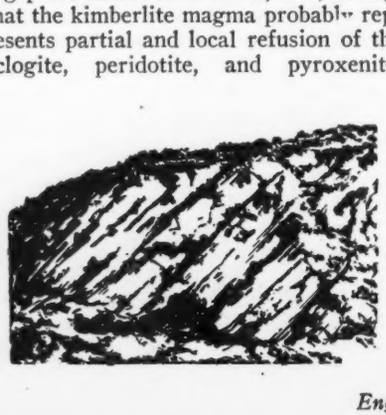
GENESIS OF THE DIAMOND. *By Alpheus F. Williams. London: Ernest Benn; 2 vols.; pp. 636; illus.; £5.*

In these sumptuous and richly illustrated volumes, the author, who for more than 30 years has been connected with De Beers Consolidated Mines, for much of the time as general manager, has presented an imposing body of facts concerning the occurrence of the diamond in South Africa. He discusses fully the character and mode of formation of the kimberlite pipes, the petrography and origin of the kimberlite, and, in very interesting fashion, the remarkable inclusions of eclogite, peridotite, and pyroxenite found in that rock. The numerous colored illustrations of these included rocks, as seen in thin section, with low magnification, are particularly fine. The significant fact that these inclusions, especially those of eclogite, contain diamonds that have crystallized with the other plutonic mineral constituents is placed beyond any doubt by the text and the illustrations.

Mr. Williams concludes that the kimberlite, crowded with inclusions, moved quietly through the pipes and solidified as a magma almost identical in composition with that which, at greater depth, crystallized as eclogite and other basic plutonic rocks. He sees little evidence of any violent or explosive action in the emplacement of the rock that now, in more or less serpentinized condition, is known as kimberlite. Concerning the actual origin of the diamonds in the kimberlite, there appears to be no clean-cut or emphatic statement, but the general tenor of the discussion indicates that Mr. Williams regards them as having crystallized from the kimberlite magma at some stage in its ascent through the pipes.

In my brief visit to the diamond mines following the session of the fifteenth International Geological Congress in 1929, I was impressed, as was Mr. Waldemar Lindgren earlier by an examination of specimens, by the obvious evidence of pyroclastic structure. The typical kimberlite of the diamond pipes is clearly an indurated agglomerate or volcanic breccia, having many points of resemblance, so far as structure is concerned, to the breccia of the Cripple Creek volcanic plug. The evidence of attrition, rounding, and even polishing of some of the fragments is unmistakable, and much of this evidence is frankly presented by Mr. Williams. He attempts to account for the rounded fragments by supposing that they underwent abrasion during their upward progress in an upwelling column of lava. Such a process, however, appears to me as highly improbable. The whole aspect of the kimberlite, even disregarding its pyroclastic character, renders improbable the view that the diamond, a mineral which there are many reasons to believe could have crystallized only under exceptionally plutonic conditions, can have originated within the particular magma represented by the kimberlite. In this connection, the undoubted occurrence of diamonds in considerable quantity in the eclogite inclusions appears highly significant. It would be truly extraordinary if the diamond could crystallize in plutonic eclogite and, at a later period or stage, crystallize again, under entirely different conditions, from a magma in which fragments of eclogite were present as inclusions. That a mineral so rare should be formed twice at any particular locality, under totally different conditions, would be an amazing and incredible coincidence.

It is with considerable diffidence that, with an experience in the diamond mines limited to a few days, I venture to oppose my view to that of a man to whose investigative spirit, industry, and long experience these volumes bear eloquent testimony. The reading of the work, however, confirms the opinions, reached in 1929, that the kimberlite (exclusive of some dikes) is essentially a volcanic agglomerate, with many foreign fragments; that it represents explosive volcanic activity; that the pipes occupy local enlargements of fissures, the initial widening having perhaps been effected by the escape of hot gases under high pressure; that these vents were finally filled, not with molten lava but with the fragmentary material, of more or less diverse origin, resulting from repeated explosive eruptions; that the diamonds originated in the eclogites and related plutonic rocks and were mechanically transported, like the eclogite inclusions, to their present resting place in the kimberlite; and, finally, that the kimberlite magma probably represents partial and local refusion of the eclogite, peridotite, and pyroxenite, which, on the evidence of inclusions, exist at some unknown depth.



In Volume II the crystallography and other physical characteristics of the diamond are fully described and finely illustrated. Here, also, is an interesting chapter on the diamond-bearing alluvial deposits of the Union of South Africa, in which is traced the long journey of some of the stones, down the Vaal and Orange rivers, to the Atlantic Ocean and the beaches of southwest Africa.

A noteworthy feature of the book is its testimony to the large amount of patient scientific work, including the study of numerous petrographic thin sections, which has been devoted to it by a busy manager of a great industrial organization, to whom such study has clearly been a source of happiness and recreation. From an editorial point of view, the work could probably be improved by the omission of certain material, particularly the long citations from easily accessible texts, as, for example, the quotation on olivine (Vol. II, pp. 363-365) from Iddings' translation of Rosenbusch. Some of the illustrations of thin sections might also be easily spared. Chemical analyses are presented in great profusion; but, unfortunately, the names of those responsible for the analytical work are not in all cases explicitly stated; although, in the preface, general credit for the chemical work is given to the late John Parry, chief chemist of the De Beers company. The volumes are beautifully printed on good paper and are remarkably free from typographical errors. Among the few noted, however, may be mentioned the superfluous second "a" in *"Chamberlain"* (cited on p. 431 of Vol. II), and, as a minor misstatement, the reference to the late Dr. George F. Becker (Vol. I, p. 309) as "George Becker, Director of the Carnegie Institute" of Washington. Dr. Becker was never director of the Carnegie Institution.

Mr. Williams is to be heartily congratulated on the publication of this monumental work, and it is to be hoped that he may continue to pursue, for many years to come, the subject to which he has made so outstanding and permanent a contribution. If I have suggested some divergence of view as to the genesis of the diamond, this expression of opinion has been offered without any pretensions to authority or finality and with appreciative recognition of Mr. Williams' genial personality and his uniformly tolerant attitude towards scientific conclusions at variance with those which he has himself reached.

THE BUSHVELD IGNEOUS COMPLEX OF THE CENTRAL TRANSVAAL. *By A. L. Hall. Memoir No. 28, Geological Survey, Union of South Africa. Pretoria. Pp. 560; plates 41; figures 41. Price 10s.*

This account of the Bushveld as a whole—a colossal task—was undertaken by Dr. Hall in response to a suggestion of Prof. R. A. Daly and his associates of the Shaler Memorial Expedition to South Africa in 1922. The delay in its appearance the author ascribes in part to various circumstances, such as the devotion of three years to preparing for the fifteenth International Geological Congress held in 1929. This delay made it possible for him, however, to draw upon the evidence disclosed by the intensive prospecting and mining work following the discovery of platinum in the Bushveld norite in 1924 and to take advantage of the criticism by many well-known geologists who attended the International Congress and of field evidence obtained on an excursion through the eastern end of the Bushveld, a trip that was among the features of that gathering. A further benefit of the delay has been that in the meantime the systematic survey of the Complex has been completed. Dr. Hall's objective has been to show the present state of existing knowledge "regarding the major incidents of the Bushveld cycle of igneous activity, to discuss its position in the stratigraphical story of the Central Transvaal, and to recognize and define the great underlying problems, in the hope that their fuller realization may suggest the most promising lines of further enquiry." He begins by giving a 119-page summary of the "entire Bushveld literature," which is indeed extensive. Chapter II is a general review of the Complex. Chapters III, IV, and V deal respectively with the face of the Complex, the rocks of the framework—that is to say, its foundations, roof and related members—and, third, the structural design. Chapters VI to X, inclusive, treat of the various phases of the Bushveld petrogenic cycle. Chapter XI deals with the metamorphic effects of the Complex in floor and roof. Chapter XII considers the satellites of the outer Complex; Chapter XIII the alkaline rocks, which, though probably later than the youngest major component of the Complex, nevertheless fall within its geographical limits. In Chapter XIV the geological history of the Complex is set forth. The next and concluding chapter summarizes the mineral resources of the Complex. The ore deposits, not always economic, which are associated with it, include those of gold, platinum, and silver; tin, lead, zinc, vanadium, chromium, iron, tungsten, molybdenite, bismuth, arsenic, cobalt and nickel; asbestos, fluorspar, monazite, barite and soda; and various quarry products, including norite, granite, syenite, South African jade, serpentinized dolomite, foyaite-porphyry, marble and limestone. Gold, it might be added in conclusion, according to Dr. Hall, has not so far proved a feature in the area's potential wealth. But, he says, "the dense cloud of international depression now resting heavily on the realization of all competitive wealth should not be allowed to obscure the fact that the Bushveld contains important resources of proved economic value, that may with confidence be expected to come into their own again."

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WHAT IS TECHNOCRACY? *By Allen Raymond. New York: McGraw-Hill Book Company; pp. 180; $1.50.*

The word "technocracy" has enjoyed a brief but hectic publicity, but since the disavowal of many of Howard Scott's ideas by most of his erstwhile associates in research, and the departure of Mr. Scott from the precincts of Columbia University, public interest in the word has subsided. The task remains for some impartial agency to make an unemotional and scientific study of disclosed facts.

This volume, now mainly of historical interest because of the swift march of events, is an account of the technocracy enterprise, as interpreted by a newspaper journalist, with emphasis on the personality and past career of Howard Scott and on criticisms of what technocracy has been interpreted to mean.

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TASCHENBUCH FÜR METALLURGISCHE PROBIERKUNDE BEWERTUNG UND VERKÄUFE VON ERZEN. *By C. Frick and H. Dausch; pp. 250; illus. Stuttgart, Germany: Ferdinand Enke Verlag; Rm. 14.40.*

Docimacy, the art of testing ores and metallurgical products, is the subject treated in this volume. Although the oldest branch of the techno-chemical analysis, it has by no means outlived its usefulness, but is still of great importance for preliminary work in the field and for the small plant where expensive laboratory equipment and specially trained personnel are lacking. In addition to the important tests for the non-ferrous materials, the book contains detailed descriptions of equipment. Of practical interest is the chapter on field work, prospecting, and scientific expeditions, in which equipment and chemicals required for the various purposes are presented in tabulated form. Other chapters include blowpipe work, analysis of gas and fuels, and sampling of ores. A large part of the book deals with customs work, and evaluation of the different types of customs ores, with detailed treatment of the important commercial considerations in this field. Numerous illustrations and many useful tables are included.

NATIONALITY OF COMMERCIAL CONTROL OF WORLD MINERALS. *Report of The Mineral Inquiry, American Institute of Mining and Metallurgical Engineers. New York: American Institute of Mining and Metallurgical Engineers. A pamphlet; pp. 44; 75c.*

As Prof. C. K. Leith—chairman of the committee composing "The Mineral Inquiry"—explains in a foreword, this report is the first of a series planned to make factual studies of the world's mineral resources in their political and international relations. The eighteen minerals discussed in the report comprise aluminum, chromite, copper, iron and steel, lead, manganese ore, mercury, molybdenum, nickel, nitrates, petroleum, potash, silver, sulphur, tin, tungsten, vanadium, zinc. Coal, although it is one of the most important minerals, is omitted, the chairman states, because not more than 1 or 2 per cent of the world's production comes from mines that are owned by companies foreign to the countries where the coal is produced. Other minerals, of which gold is typical, are omitted because sufficient data have not yet been accumulated; some of these will probably be covered in subsequent reports.

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MINERAL RESOURCES OF THE UNITED STATES, 1930. *Part II—Non-metals; pp. 876. By O. E. Kiessling. Washington, D. C.; U. S. Department of Commerce, Bureau of Mines; $3.*

This statistical volume of the U. S. Bureau of Mines for the year 1930 covers the identical ground surveyed in the 1929 volume issued by the same agency. Eighteen more pages are contained in the later work, and the former detailed summary of contents is condensed to a listing of the thirty subjects that comprise the Bureau's classification of the country's non-metallic industries.

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LA GÉOLOGIE ET LES MINES DE LA FRANCE D'OUTRE-MER. *By L. Joleaud, P. Fallot, L. Neltner, and others. Paris, France: Société d'Editions Geographiques, Maritimes et Coloniales; pp. 604, illus.*

This volume, constituting many of the lectures delivered by the authors at the Muséum d'Histoire Naturelle in Paris during the winter 1931-32, is a symposium on the geology and mining status of the French colonies, including such little-known regions as the Kerguelen, the Terra Adélie, and Clipperton Island. It is well illustrated by maps, some in two colors. Among the authors are to be noted the names of M. Lacroix, who contributed the section dealing with the crystalline formations of Madagascar; M. Jacob, chief of the Geological Survery of Indo-China, and MM. Joleaud and Bertrand, professors of the Sorbonne. Each chapter is followed by a short but selected bibliography.

DEVELOPMENT OF AMERICAN INDUSTRIES. *By J. G. Glover and W. C. Cornell. New York: Prentice-Hall; $5.*

This volume constitutes a symposium, by forty-one contributors, covering thirty-nine industries, with additional articles on labor and trade associations. The articles on copper, lead, and zinc have been contributed by W. A. Willis, manager, Copper & Brass Research Association; F. E. Wormser, secretary and treasurer, Lead Industries Association; and J. D. Conover, secretary, American Zinc Institute, respectively. Excerpts follow:

"Since the earliest recollection of man, copper, brass, or bronze has had a part in our scheme of life; but it has been only in recent years that its field has been broadened to embrace almost every phase of our existence. . . . The collapse of the Central Powers in the recent world conflict was due as much to their shortage of copper as to any other one factor. . . America leads the rest of the world in the production of copper. . . . Figures for 1929, the peak year in American copper production, that of the United States amounted to 1,026,348 tons, or approximately half of the world's total production. . . . Just around the corner is a new development in architecture and building which is expected to consume vast quantities of copper. . . . The all-copper house which already is in service in Germany. The principle involved is the pre-fabrication of units to be assembled on the building site."

"The production of lead from American mines is roughly about one-third of the total world production, a proportion which exerts an extremely important influence upon the world market but does not, and cannot, control it. . . . Unlike the copper miner, the lead producer does not depend on the export market to take a part of his product. . . . Since 1921 lead has definitely drawn away from its dependence upon the white lead industry—and consequently the paint business—to furnish its most important market, and has turned to two essentially electrical applications for its principal consuming outlets, storage battery manufacture and cable covering. It is fortunate for lead that these two relatively newer uses have grown rapidly in recent years, for the white lead market has gradually declined owing to intense competition from other paint pigments. . . . At the present time a factor of great moment in the lead situation is the growing importance of secondary, or scrap, lead as a source of raw material for the manufacturer. . . . For every three tons of new lead mined, it is believed that about one ton of old lead is reclaimed."

"Due to the high grade of many zinc deposits in foreign countries and the low cost of production under modern metallurgical treatment, these foreign countries are now supplying the world market with an overabundance of low-priced zinc. . . . The American industry has been compelled to devote itself mainly to the domestic market. There are large reserves of zinc ore in the United States which are capable of meeting all probable domestic demands for many years to come."

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TEMPERATURE AND HUMIDITY MEASUREMENT AND CONTROL. *By M. F. Béhar. Pp. 420; illus. Pittsburgh, Pa.: Instruments Publishing Company; $2.*

This handbook constitutes Parts II and III of the Manual of Instrumentation that Mr. Béhar is preparing. Much of the contents has appeared serially in *Instruments*, but even this material has been largely revised. Besides its excellence as a summary of temperature control and pyrometrical technology, the volume is of particular value because of the comprehensive discussion of humidity. Part III, dealing with this subject, consists of four chapters devoted to general facts concerning humidity, humidity measurement, humidity and aerological measuring instruments, automatic control of humidity—a part of the book that will undoubtedly have most appeal for the mining engineer, because underground projects afford extensive opportunity for the application of air conditioning. Operating efficiencies have been materially improved in other industries by air conditioning, with both direct and indirect benefits accruing to employer and employee. Similar results may reasonably be expected in the field of mining, and Mr. Béhar's work would appear to be an admirable aid in achieving them.

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ELEMENTS OF MINING. *By George J. Young. Third Edition. Pp. 707; illus. New York: McGraw-Hill Book Company; $6. London: McGraw-Hill Book Company, Ltd.*

This, the third edition of a standard textbook, retains its original form and serves its original purpose—to provide the student of mining with a starting point of fundamental theory and fact. A thorough revision has been necessary to bring it up to date, and much new material has been added to illustrate recent developments in methods and equipment, to reduce operating cost, and to increase technical efficiency. Although of especial value to the mining student, and to the teacher of mining engineering, the new edition will prove a useful working tool for the mine manager and operator who requires accurate and easily accessible information on every phase of ore extraction, and of the preliminaries and incidentals to the development and operation of a mining property of any type.

ELEKTRIZITÄT UNTER TAGE. *By. W. Philippi. Leipzig, Germany: S. Hirzel. Pp. 188; illus.; price, M. 17.40.*

This monograph, the ninth in a series on industrial applications of electricity, covers the field of underground mining. No attempt has been made to give an exhaustive description of the apparatus involved, and the mining engineer with an average knowledge of electrical science should have no difficulty in assimilating the contents. Treatment includes a general description of the common types of motors, generators, transformers, and converters; underground cables and transmission lines; hoisting and pumping machinery; drilling and cutting equipment; scraper hoists and various types of conveyors; and underground haulage.

The discussion of underground lighting arrangements—the ordinary types as well as flood lighting—deserves special attention. Tests at the Wenzeslaus mine, in Lower Silesia, have shown an increase of about 25 per cent in the tonnage handled per man-day after substituting stationary electric lights for the portable lights formerly used. Another interesting feature was the reduction in the amount of waste trammed—about 50 per cent.

Economic considerations and safety devices have also been discussed. Excellent illustrations and sketches add to the value of the book.

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GEOLOGY OF SOUTHWEST ANKOLE AND ADJACENT TERRITORIES, WITH SPECIAL REFERENCE TO THE TIN DEPOSITS, 1932. *By A. D. Combe. Appendix on the petrology, by A. M. Groves. Memoir No. II, Geological Survey of Uganda, Entebbe, Uganda. Price 35s.*

This memoir consists of three parts, the first two dealing with the geology of part of Southwest Ankole and Rukiga and with that of the Bukoba Province of Northwest Tanganyika Territory, respectively. Part III, on economic geology, is of principal interest. The probability that tin would be discovered in Ankole was responsible for hastening the mapping of these regions. Cassiterite is the only mineral of economic value found in workable quantities in the area covered by the memoir. It is reported as occurring in quartz veins and in masses and veins of muscovite and hydrous micas associated with them, and in quartz-muscovite-pegmatite veins, detrital, eluvial, and alluvial deposits. The quartz veins and masses and veins of micas associated with them have been proved the most important from the economic point of view. Intensive prospecting was still being carried out when the report was written (it is dated 1932), at which time the only mine maintaining regular production was that of Kagera Tinfields, Ltd., at Mwirasando, in Uganda.

Personal Notes . . . Comment . . . Criticism

Dr. Frederick M. Becket, recently elected president of the A.I.M.E. for 1933-34, is of Canadian birth, and was educated at McGill and Columbia universities. He has made significant contributions to high-temperature metallurgy, and was awarded the Perkin medal in 1924 for distinguished service in applied chemistry. He was president of the American Electrochemical Society, 1925-26, and is now president of Union Carbide & Carbon Research Laboratories.

Frank C. Hooper, manager of Barton Mines Corporation, producing garnet at North Creek, N. Y., and inventor of the Hooper pneumatic jig, is reported to have discovered a deposit of a rare mineral, serendibite, in the Adirondack Mountains west of Mill Creek in the town of Johnsburg, Warren County, N. Y. This mineral has been previously known to occur only at Gangapitiya, Ceylon. At Johnsburg an occasional narrow band of it is found in limestone or quartz.

Charles Hardy has returned to New York from Europe.

R. W. Howe, of Montreal, has completed an examination of the Moss mine, in northwestern Ontario, for Montreal clients.

Prof. W. H. Haas is serving as acting chairman of the department of geology and geography at Northwestern University.

W. E. MacBoyle, and his sons **M. D.** and **A. D. MacBoyle,** mine operators of British Columbia, have been visiting mines in Mohave County, Ariz.

James O. Elton, recipient of the James Douglas medal at the recent meeting of the A.I.M.E. in New York, was educated at Washington State College and the University of Idaho. Association in a metallurgical capacity with Anaconda Copper was followed by appointment as manager of International Smelting and its subsidiaries, a position held since 1921. The citation accompanying the award pays tribute to Mr. Elton's contributions to the problem of smelter-fume treatment and electrolytic zinc production, to his success in the development of commercial methods for preferential flotation of lead-silver-zinc ores and oxidized lead-silver ores, and in improving the technique of lead smelting.

H. J. C. MacDonald is now manager of mines and mills for the Glavsvetmetzoloto—the organization that directs the non-ferrous metal industry of the U.S.S.R.

Sydney H. Ball, recently elected president of the Mining & Metallurgical Society of America, is a consulting engineer with headquarters in New York, well known to readers of *Engineering and Mining Journal* as an international authority on the geology and mining of diamond deposits. He was graduated from the University of Wisconsin, on the faculty of which he subsequently served. Experience on the staffs of the Missouri Bureau of Mines & Geology and the U. S. Geological Survey was followed by professional work in the Belgian Congo, Europe, Borneo, and the Americas.

Dr. F. L. Ransome. who contributes to this issue a critical study of the most recent treatise on the origin of the diamond, is professor of economic geology at the California Institute of Technology and consulting geologist to the Metropolitan Water District of Southern California and the U. S. Bureau of Reclamation. He has served on the faculty of Harvard University and the University of California, and on the staff of the U. S. Geological Survey, and as consultant to the War Department and to various private enterprises.

Alan Anderson has been appointed manager of the Central Patricia property, northwest Ontario.

James A. Potter, formerly superintendent of the New Cornelia mill at Ajo, is in charge of the Best Bet property in the Chemehuevis Mountains, 40 miles southwest of Yucca, Ariz., owned by Mrs. Isabella Greenway.

E. L. Brown, who developed the Sheritt-Gordon mine, has been engaged by God's Lake Gold Mines to develop its property—the most remote gold mine now being opened in Canada. The enterprise is under the general direction of **R. W. Jowsey.**

A. J. McDermid, who contributes the article on the processing of feldspar in this issue, is a graduate of the Minnesota School of Mines who specialized in geological work in the Mid-Continent oilfields prior to War service. Subsequently he served on the staff of several Western mining companies, and since 1923 has supervised mine contracts for Miami Copper, at Miami, Ariz.

W. H. Aldridge, recipient of the A.I.M.E. W. L. Saunders medal for achievement in mining, was graduated from the School of Mines of Columbia University in 1887, and practiced his profession in Colorado and Montana until 1897, when he became associated with the Canadian Pacific Railway, established lead smelting and electrolytic lead refining in Canada, and organized the Consolidated Mining & Smelting Company. He became managing director of Inspiration Copper in 1911, president of Texas Gulf Sulphur in 1918. He is a member of the Mining & Metallurgical Society of America and the Institution of Mining & Metallurgy.

Heath Steele, who is in Mexico inspecting the properties of Cia. Minera de Peñoles, will return to New York late in March.

D. C. Jackling will receive the honorary degree of Doctor of Engineering from the Missouri School of Mines and Metallurgy in May next.

L. L. Bolton, Assistant Deputy Minister of Mines at Ottawa, recently summarized the radium-silver development at Great Bear Lake in a radio address over a Canadian circuit. During less than three years since **Gilbert Labine** made the first discovery, a considerable tonnage of rich ore has been developed, high-grade silver ore has been smelted, and a plant to treat the pitchblende has been established in Ontario, using a process worked out by **R. J. Traill,** of the Department of Mines at Ottawa.

F. S. McNicholas, author of the article in this issue on mining caved areas, a graduate of the Colorado School of Mines, has served in various capacities on the staffs of Timber Butte Milling, Montana; Anaconda Copper, Montana and Idaho; Butte & Superior, Montana; Buena Vista Mining, Cuba; Bar Principal Leasing, Colombia. Since 1929 he has been mine superintendent in the Hidden Creek property of Granby Consolidated Mining, Smelting & Power, British Columbia.

H. W. Gould, of San Francisco, was recently in Alabama and Georgia, examining gold deposits.

W. H. Trewartha-James is in Australia.

C. O. Stee, manager of Siscoe Gold Mines, operating at Siscoe, Que., reports a decision to increase mill tonnage from recent capacity of 180 to 225, and later possibly to 300 tons per day. Flotation will follow the blankets; the concentrate produced will be reground and sent to the cyanide unit. The change-over was effected late in February. Discharge from the ball mill will pass to hydraulic concentrators, where a concentrate containing heavy free gold and about 80 per cent of the gold in the ore will be removed. Ratio of concentration at this point will be about 100:1. In closed circuit with the ball mills will be Dorr classifiers, through which the overflow from the hydraulic concentrators passes. Overflow from the classifiers will go to blanket tables, using a heavy corduroy of the type developed in South Africa and now used in several Canadian mills. These blankets will recover an additional 5 to 7 per cent of free gold. Tailing from the blanket tables will be sent to flotation machines by way of a conditioner. Concentrate will be dewatered, reground, and sent to the cyanide unit. The recovery expected will be about 98½ per cent.

E. A. Montgomery, who recently purchased the Keystone mine on the Silver City branch of the Comstock Lode, in Nevada, was the discoverer of the Montgomery-Shoshone mine at Bullfrog, in 1905, which he sold to **Charles M. Schwab.** In 1907 he developed the Skidoo mine in an almost inaccessible canyon in the Funeral Range, overlooking Death Valley, in California, and built a 100-ton mill there.

Morris J. Elsing, formerly geologist to Copper Queen Consolidated, later in charge of the geological department and of the mining operations of Cananea Consolidated, commences in this issue a new series of articles on the cost of ore transportation in mines.

Ely C. Hutchinson, formerly president and general manager of the Pelton Water Wheel Company, of San Francisco and New York, has resigned as editor of *Power* to accept the position of president of the Edge Moor Iron Company, Edge Moor, Del.

Arno S. Winther, general manager of Rhokana Corporation, Northern Rhodesia, recently visited **Frank A. Wardlaw, Jr.,** general superintendent for Inspiration Copper, at Inspiration, Ariz. Mr. Winther left Rhodesia on a vacation in November, 1932, and expects to return there in March.

Dr. M. J. Millzner, in charge of an emergency hospital at San Francisco, Calif., is reported to have injected 50 cc. of an aqueous 1 per cent solution of methylene blue as an antidote for poisoning by potassium cyanide (15 grains), with complete success.

Lewis W. Douglas, son of James S. Douglas, president of United Verde Extension, grandson of the late James Douglas, has been appointed Director of the Budget by President Roosevelt.

J. T. Couchman has been appointed manager of the King Cassilis mine, near Omeo, Vic., Australia.

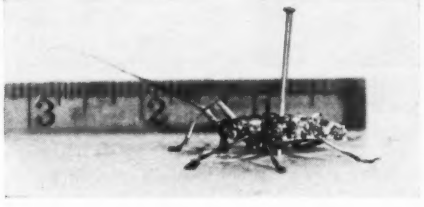
George H. Dern, formerly Governor of Utah, at one time treasurer of Mercur Gold Mining & Milling, general manager of Consolidated Mercur Gold Mines, co-inventor with T. P. Holt of the Holt-Dern chloridizing roast-leach process for the treatment of low-grade silver-lead ores, organizer of Tintic Mining, and consulting metallurgist to Tintic Standard, has been appointed Secretary of War by President Roosevelt.

G. W. Gray, technical director on the staff of Rio Tinto (Spain), recently elected president of the Institution of Mining & Metallurgy, was graduated from the Royal School of Mines in 1899. He has had a wide geographical experience, having served in various techno-executive capacities with Panuco Copper and Refugio Syndicate (Mexico); Mountain Copper (California); Loemar Mining (Borneo); Kyshtim Mining (Russia); and Pyrites Corporation (Virginia).

Lead-Boring Beetles

The Editor:

Inclosed please find a photograph of a *Monochamus Titillaria,* commonly known as a lead-boring beetle. This beetle and several hundreds of his kin did much damage to the cell sumps in the Evans-Wallower Zinc Company's plant at East St. Louis. I was fortunate enough to secure this one and have it photographed, and Mr. Saterthwaite, of the U. S. Department of Agriculture, identified it for me.



These beetles were in the Southern yellow pine used in the sump construction. Mr. Saterthwaite told me that they were deposited there by the female beetle to a depth of from 1 to 6 in. from the bark of the tree. When these eggs hatched, the worm first formed eats the surrounding wood and then goes to sleep, later developing into the beetle shown in the photograph, which bores a way to freedom in a straight line. Some of them bored diagonally through a 10x10 in. timber, then through the 3-in. siding of the sumps, and finally through the 9-lb. lead sheeting lining the sump. I measured the depths of many of these holes and found several from 14 to 16 in. long, with an average diameter of ⅜ in.

Mr. Saterthwaite advised me that these beetles might hatch from one year to twenty-five years from the time they were placed in the wood by the female, and that the only way he knew of getting rid of them was to heat the sumps with live steam until the temperature of the wood had been raised to a point that would kill all the larvæ. This method, though expensive, was adopted; and for six months from that time on there were no evidences of their activity.

HENRY P. EHRLINGER.

Warsaw, Ind.

Obituary

John D. Ryan, chairman of the board of Anaconda Copper, Chile Copper, and Chile Exploration, president of United Metals Selling and Montana Power, director of National City Bank and other corporations; at New York on Feb. 11, of heart disease; aged 69. Although the son of the discoverer of what are now known as the Copper Range mines in the Lake Superior region, Mr. Ryan elected to work in a small store at seventeen, in Michigan. At 25 he became a salesman for a lubricating company in Colorado, a position held until he was 32. Contact with Marcus Daly, promoter of the Anaconda organization, then led to acquisition of stock in the financier's banks, until control was secured. Mr. Ryan then obtained management of the business of Amalgamated Copper in Montana, leading to the presidency of the company. After the dissolution of Amalgamated Copper, and its absorption by Anaconda Copper, Mr. Ryan became president of the latter. He created the Montana Power company and served as Assistant Secretary of War, and chairman of the Aircraft Board, under President Wilson.

✦

Benjamin B. Thayer, first vice-president of Anaconda Copper, president of Raritan Copper Works, International Lead Refining, Electrolytic Zinc Process, and director of many other mining and fabricating companies; at New York on Feb. 22, after an operation; aged 70. Mr. Thayer was graduated from the Lawrence Scientific School of Harvard University in 1885, and gained practical experience in mining in South Dakota, Butte (Amalgamated Copper), the Coeur, d'Alenes, and California. In 1904 he reentered Amalgamated Copper, and was later made president of Anaconda Copper. When Amalgamated Copper was dissolved in 1915, John D. Ryan was reappointed to the presidency of Anaconda Copper, Mr. Thayer becoming vice-president in charge of mining operations. He served on the Naval Consulting Board during the War, and was president of the A.I.M.E., 1914-15, and of the Engineers' Club, New York, 1925-26.

✦

Daniel J. Williams, nephew of David Lloyd George, manager of the Hirst-Chichagof mine, Alaska; on Jan. 8, at the mine. Mr. Williams, who was born in Wales, emigrated to the United States at an early age, and served an apprenticeship in the Butte mines, ultimately graduating from the University of Montana. Prior to taking charge of the Hirst-Chichagof property Mr. Williams was manager of a gold mine near Hazelton, B. C.

✦

C. M. Knatchbull-Hugessen, fourth Baron Brabourne, chairman of Consolidated Gold Fields of South Africa; at sea, while en route from London to South Africa, on Feb. 16; aged 69.

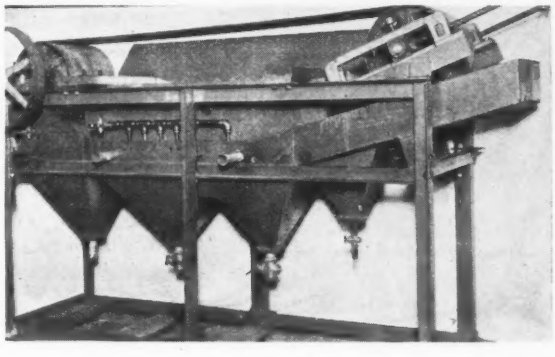
✦

Robert Henderson, "Klondike Bob" of 1896, starter of the Klondike rush of 1898; at Vancouver, B. C., on Jan. 24; aged 78.

INDUSTRIAL PROGRESS

Tests With New Wet Belt Magnetic Separator Show Interesting Results

The so-called Crockett-type wet magnetic iron ore separator, made by Dings Magnetic Separator Company, Milwaukee, was described briefly in *Engineering and Mining Journal*, October, 1932, on page 544. This separator is of the submerged-belt type, the separation of the ore mineral being effected under water. Additional information regarding it has been supplied by G. I. Thatcher, district manager of the Dings company. Its features may be summarized as follows:



Wet magnetic ore separator of submerged belt type, installed in concentrator of Alan Wood Mining Company, Dover, N. J.

1. A single unit can produce a clean tailing, a true middling, and a high-grade concentrate, making simplification of flowsheet possible.
2. Capacity is large.
3. Unsized ores from 4 mesh down to the finest dust can be handled.
4. A particular ore can be concentrated at the maximum grain size possible for that orebody.
5. Concentrates are dewatered sufficiently before discharge to make a dewatering device unnecessary. They can be sintered at minimum cost.
6. Supervision, attendance, and maintenance are reduced to a minimum; likewise the water and power required.

Allowable tailing loss has been set arbitrarily at 1 per cent magnetic iron. Concentrate grade, however, is dictated by the ore market and the separator has therefore been designed so that the grade can be varied as desired by throwing into the middling made varying amounts of true middling, or those grains which have not been crushed sufficiently to break the iron mineral from the gangue.

As the grain size decreases, wet magnetic separation has an advantage over dry in that the electrostatic force between particles disappears, mass magnetization is lessened, and the water acts as lubricant between particles, facilitating the elimination of the gangue.

The new separator has been developed to concentrate magnetite under the following conditions: (1) Where wet concentration is desirable; (2) where the size of the ore grain requires crushing to 4 mesh or finer; (3) where concentrate grain size must be kept at a maximum; (4) where phosphorus must be removed; (5) where finely ground ore is concentrated and floor space must be conserved. (A so-called 36-in. machine is 6 ft. wide over all, 6 ft. high and 9 to 15 ft. long, depending on the ore and on the degree of concentration required.)

In operation, the ground ore is fed wet at one end through a trough. The tailing is discharged through the second spigot with about two-thirds of the water, the middling through the third spigot with most of the remaining water, and the concentrates, after being carried up out of the water, through the last opening.

Separation of the tailing is effected by passing the feed through a strong magnetic zone of special shape, sufficiently agitated to overcome mass magnetization and to keep strings from forming. The tailing is carried away by the water and the iron-bearing material by the belt. Belt speed has little effect on this separation.

The volume of middling and concentrate is now presented to a zone where the middling is separated with the aid of the factors of belt speed, specially designed alternating pole magnets, "winnowing" action of the ore, and directed water currents. The middling is carried away with the water used as was the tailing. The concentrates are then carried out of the water and discharged.

Ores that vary widely in grain structure, size of ore crystals, and iron and phosphorus content have been tested with the results given in the accompanying table.

Tests Nos. 23 and 24 were run to show the relative concentration and the elimination of phosphorus secured as compared with the results of the present practice of a dry concentrator. Sample 23 consisted of mill concentrates which were concentrated without further grinding. Sample 24 was crude ore ground to the same size and concentrated. The results show that this ore will give about a 66.5 per cent iron concentrate when ground to pass 8 mesh, irrespective of amount of gangue in the crude.

The designers believe that the design favors the elimination of phosphorus in those ores where it occurs as a separate non-magnetic grain and the grinding is such as to free the phosphorus grains from the magnetic iron-ore grains. Where the ore is ground sufficiently to produce a clean tailing and a concentrate before going to the magnetic separator, only two products would be made—namely, a concentrate and a tailing.

Power required is about 3 kw. total for a machine having a capacity of 20 tons per hour feed.

The amount of water used with Samples 23 and 24 was 2.07 lb. per pound of ore. This ratio would increase somewhat with finer grinding to about 3 lb.

Table I—Performance Data

Test Number	Size of Feed	Feed Tons per Hour per Foot of Belt Width	Magnetic Fe in Feed, Per Cent	Soluble Fe in Concentrate	Elimination of Phosphorus in Concentrates, Per Cent
23	—8	6	61.0	66.92	....
24	—8	6	50.5	66.43	....
30	—8	6	49.0	68.10	83.8
35	—10	6	26.6	63.24	84.2
37	—48	3	22.15	66.58	....
40	—4	6	53.69	66.38	72.6
41	—65	6	54.50	68.75	88.6
42	—65 50%–200	3	42.0	66.50	....

All tailing contained less than 1 per cent magnetic iron ore as tested by Davis magnetic tube tester.

of water per pound of ore with minus 100-mesh feed.

The only moving parts are the pulleys and belt. The belt is run loosely and should have an exceedingly long life. Clean water gets under it and acts as a lubricant. The machine is designed so that the belt, which is of the ordinary 2- or 3-ply conveyor type, can be removed, replaced, and the joint vulcanized in one hour by two men.

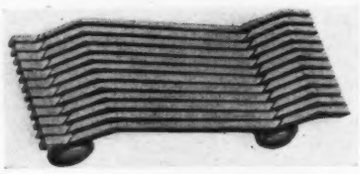
Although the details of design of the magnets are new, the construction used is the same as that which has proved to be as near permanent as any electrical machinery can be made. No repairs should be necessary for years. The magnets are stationary and thoroughly waterproofed.

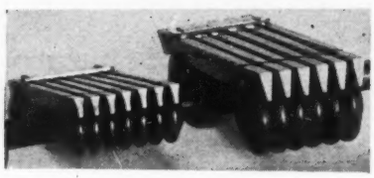
Adjustments of the machine that may be required are simple and not delicate. Once set, the machine requires practically no supervision or attention. The minimum water required is only that needed to feed the ore and remove the middling and tailing. Adding more water does not affect the process within wide limits. The speed of the belt is not a critical factor and can be varied considerably without materially affecting results.

R. E. Crockett, superintendent of Alan Wood Mining Company's Scrub Oaks mine, Dover, N. J., assisted in designing this machine. He also outlined and ran tests on the different ores from an operator's viewpoint, so as to assure the results could be duplicated in a concentrator.

Wedge-Wire Screens in Many Profiles

New screening equipment is being introduced in the United States by Abbe Engineering Company, New York. Figs. 1 to 4 illustrate some of the principal items. Fig. 1 shows the "Rima" wound wedge-wire slit sieve in two of the various profiles (or cross-sections) in which it is made. This sieve is also made in stepped form as in Fig. 2. Width of slits is from 1/500th in. and up. Spacing is uniform and is kept so by spacing lugs which prevent side movement of the wires. The screens are rigid.





Figs. 1 and 2—Sections of wedge-wire slit sieves having wires of different profiles. Above, A slit sieve in stepped form.

Screen cloth reinforced with wire, which maintains its rigidity, prevents mesh enlargement, and prolongs the life of the screen, is also made. Spacing of the wires varies with the mesh.

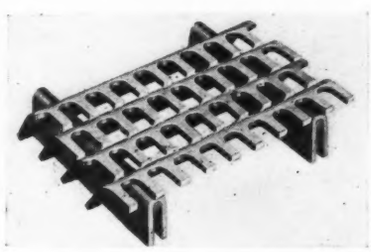


Fig. 3—Washing grate of rigidly arranged slotted steel bars.

This is intended for all vibrating, shaking and rotary screens, filter drums and dust jackets.

A washing grate (the "Denta") for jigs and washing boxes is shown in Fig. 3. It is made of rigidly arranged slotted steel bars. It is said to offer maximum open area and minimum resistance to the washing water. Claim is made that it will not choke and cannot break.

Light-weight screen cloths ("Vibro") of tempered spring or rustless steel are being made in meshes from $\frac{5}{8}$-in. hole to 60 mesh. They are made in three wire gages for each mesh. To give greater vibrating action, the wires for warp and weft are of different gages.

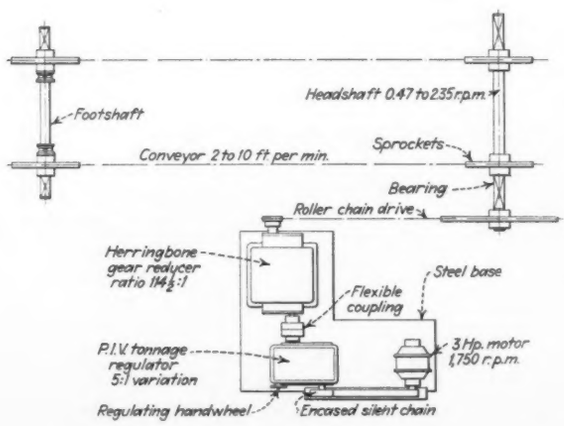
For Flotation Test Work In the Field

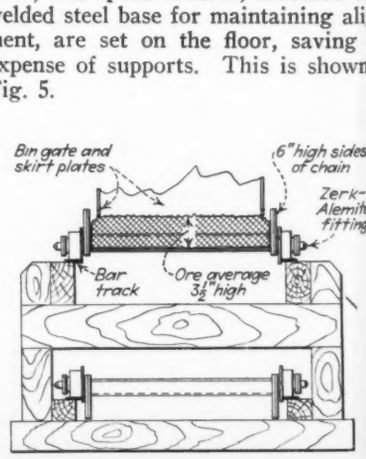
A small flotation unit for general test work in the field has been developed by Denver Equipment Company, Denver, Colo. It will handle 4- to 5-lb. batch samples. It consists of a 2,000-gram Denver "Sub-A" (Fahrenwald) cell driven by a $\frac{3}{4}$-hp. gasoline engine. A complete set of reagents for testing gold ores or placer sands is supplied with it.

An Efficient Ore Feeder

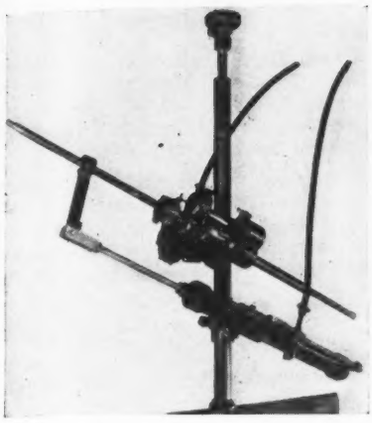
R. P. Akins, designing engineer, Denver, is responsible for an interesting feeder installation in a mill in the Western United States, according to C. M. Schloss, of Lindroot, Shubart & Company, Denver. A Link-Belt corrugated steel apron conveyor 24 in. wide, 14 ft. c. to c., feeds 3.33 to 17.5 tons per hour of minus $\frac{1}{2}$-in. ore from a fine-ore bin. The conveyor chain consists of two strands of 9-in. pitch steel-bushed steel-bar roller chain with $\frac{3}{4}$-in. diameter pins at articulations, drilled and fitted with push-type Alemite-Zerk fittings; inside chain links, forming the sides of the conveyor, are 6 in. high. The high sides prevent ore falling off the edges so that stationary skirts are necessary only at the bin opening. The skirt boards project close to the apron, between the sides; the bin gate levels the ore to a height just below the top of the sides. The result is that the cross-section of ore is absolutely uniform. See Fig. 4. Changing the speed of the conveyor apron changes the volume of the ore in direct proportion.

A silent chain drive connects the 3-hp., 1,750-r.p.m. motor to the constant-speed shaft of a P.I.V. variable-speed transmission—the tonnage regulator—whose variable-speed shaft can be regulated from 1,620 r.p.m. to 324 r.p.m. in an infinite number of steps by merely turning a hand wheel. This shaft is connected through a flexible coupling to the high-speed shaft of a herringbone-gear reducer, ratio $114\frac{1}{2}$ to 1; a roller chain drive connects the reducer slow-speed shaft to the headshaft of the feeder. Motor, P.I.V. tonnage regulator, and speed reducer, mounted on a welded steel base for maintaining alignment, are set on the floor, saving the expense of supports. This is shown in Fig. 5.





Figs. 4 and 5—Left, Sketch showing arrangement of ore-feeder drive using a gear reducer, a P. I. V. variable speed transmission and a silent chain. Right, A cross-section of feeder bed.



One-Man Core Drill Aids Prospecting

A one-man diamond core drill (No. 6) has been introduced by Sullivan Machinery Company, Chicago. It has a heavy double-grip clamp, and may be set up quickly on a mining column or cross bar. It uses standard diamond drill fittings, size "E" ($\frac{15}{16}$-in. core) or "EX" ($\frac{7}{8}$-in. core), and will accommodate fittings through the swivel head, $1\frac{1}{4}$ in. O.D. It has a capacity of 50 to 250 ft. in depth and will drill at any angle from vertical to horizontal. It weighs 207 lb. complete. The swivel head weighs 127 lb. and the compressed air "Turbinair" motor 80 lb.

The company has also developed a pneumatic rod puller for withdrawing rods from down holes, when the core drill has no hoist. As shown in a cut, the puller consists of a cylinder and piston with a valve and pipe connections to admit compressed air to either side of the piston, as desired. To the other end of the piston rod is screwed a fork, and in this is loosely held, at right angles, an arm carrying a handlike grip which encircles the drill rod. A working stroke of 37 in. is available. The puller can handle up to 370 ft. of Sullivan "E" rods, or 330 ft. of "A" rods. The puller is 55 in. long over all and weighs 114 lb. net and the saddle weighs 34 lb. additional net.

Paints Wet Surfaces

For painting wet surfaces, such as presented at times by tanks, roofs, and other equipment and structures, Quigley Company, Inc., New York, has developed an interesting aluminum paint consisting of aluminum powder in a black vehicle made from coal tar which appears to have unusual protective power. The naphtha vehicle has the property of displacing any water on the surface of the metal and of penetrating through any rust that may be in the pits on it, thus getting next to the metal itself. The aluminum powder carried in suspension then produces a superior aluminum coating, with all its decorative and protective advantages. This paint is known as Triple-A No. 44 Aluminum Vehicle.

New Transport Units

During 1932 several new types of material-handling units for mine and quarry were developed by Easton Car & Construction Company, New York. The company's new Phoenix truck body was shown in illustration in the January, 1933, issue, as in service in the quarry of the Hudson River Stone Corporation. It has 10-cu.yd. capacity. Floor and sides are lined with $\frac{1}{2}$-in. manganese steel liners, protected with an oak cushion, 2 to 3 in. thick.

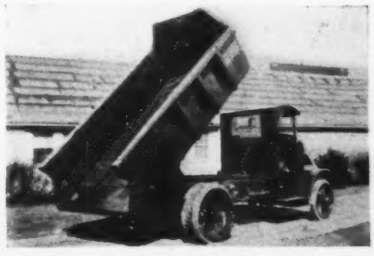
A cut on this page shows the Easton Alpha skip-type car designed to dump into crusher or hopper without interference from the door, which operates with the bail. This is particularly adapted for work on an incline, although it can be used on a level haul.

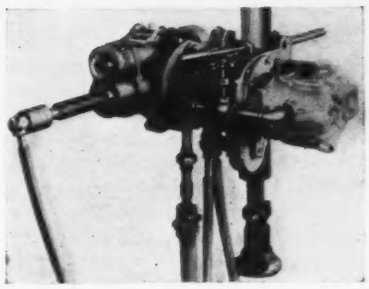
A self-propelled car operated by one man, the Lococar, having a three-way hydraulically dumped body, is shown in another cut. It has 12-cu.yd. capacity and is recommended for stockpiling, waste disposal, and interplant transport. The model shown has two 50-hp. 250-v. d.c. axle-mounted motors. Free-running speed is 16 m.p.h.

Skip-type car for level or incline haul

Motorized three-way dump car

A one-way truck body (Wonway), of 5- to 10-yd. capacity, is shown below. It has an automatic door which opens out and down. It is dumped by a hydraulic self-contained underbody hoist.





An Improved Drifter

A light, all-purpose drifting drill, of the $2\frac{1}{2}$-in. cylinder class, has been developed by Gardner-Denver Company, Quincy, Ill. Weight with standard mounting is 154 lb. It operates either wet or dry and with either round lugged or straight shank tappet steel. The separate cylinder bushing has been eliminated and a renewable bronze liner pressed into the cylinder supporting the front end of the hammer. The valve is of the tubular type.

Pumping Dry Pulverulent Materials

A device known as the Fluxo pump and intended for transporting such pulverulent materials as cement, dry clay, feldspar, gypsum, and hydrated lime, has been introduced by F. L. Smidth & Co., New York. This pump consists of two tanks mounted on a carriage so that the unit can be wheeled either on tracks or directly on the floor to a point alongside the silos or storage tanks, and the pump tank connected by means of a flexible hose with any silo from which it is desired to draw the stored material. Operation is simple, consisting of alternately subjecting the tank to vacuum and pressure. While the material is being drawn into one tank by suction, it is being expelled from the other by compressed air. Then the cycle of operation is reversed. The repetition is automatic once the machine has been started.

Where conditions permit the material to flow by gravity from the silo into the pump tank, a modified form known as the gravity feed Fluxo pump is used. Aside from the gravity feed, this operates on the same principle as already described.

Power Drive Interests Study Field

An investigation of the mechanical power transmission field has been undertaken by a group known as Mechanical Power Engineering Associates. Among the members are: Graton & Knight Company, the American Pulley Company, and SKF Industries, Inc. Charles E. Brimley, president of American Pulley, is president of the new group. The purpose of the investigation is to ascertain the most efficient form of power transmission, whether group or individual drive, for individual plant conditions. A series of case studies has been made.

Pumping Unit Automatically Primed

A new pumping unit introduced by Worthington Pump & Machinery Corporation, Harrison, N. J., is said to make possible effective remote control whenever suction lift exists and to eliminate costly and inaccessible wet and dry pit installations and foot valves. It is also said to prevent air binding, to overcome loss of suction, and to make ejectors, priming tanks, or hand primers unnecessary. It comprises an electrically driven high-efficiency ball-bearing centrifugal pump, mounted with its motor on a fabricated steel bedplate, together with a priming unit of the wet vacuum type controlled by an electric pressure switch. The priming pump or evacuator is an adaptation of the Nash Hytor pump, which has been licensed to Worthington. The new unit is now available with certain sizes of Worthington centrifugal pumps, but will soon be available with any type or size of the company's make.

Anti-Friction Conveyor Carrier

A new anti-friction conveyor carrier has been developed by Barber-Greene Company, Aurora, Ill. It has self-aligning Shafer roller bearings which take thrust as well as radial load. Each size has but one size roller unit, and the units have component parts, thus being interchangeable. The carrier can be interchanged, moreover, with any standard carrier. The roller is of 5½-in. diameter seamless tubing made of No. 9 gage metal. Ends are of pressed steel with the outer roller race inserted under pressure. The inner tube is 1⅛ in. in diameter, with two shoulders, one at each end. The spindle is ¾ in. in diameter and of selected cold-rolled steel drilled and tapped for pressure fittings.

Motor Starters Use New Contactors

General purpose mill-type starters for constant-speed d.-c. motors have been introduced in a new line ranging from 1 to 75 hp., by the General Electric Company. They utilize newly developed accelerating contactors designed to act as a relay to provide time delay and to short out the starting resistor by means of a contact finger. This contactor thus replaces two conventional devices: the accelerating relay and the old accelerating contactor.

Time Limit Starter Non-reversing

A new time starter has been designed by Westinghouse for mining service for use with non-reversing, constant-speed direct-current motors on pumps, fans, hoists, conveyors, loading machines, and other equipment. In it, by use of a recently developed device known as the "Timetractor," a definite time-limit acceleration is obtained without use of intermediate relays.

Corrosion Resistant Lining for Tanks

Extension of the use of steel tanks lined with Triflex rubber lining which is protected by brick sheathing is suggested by the B. F. Goodrich Rubber Company, Akron, Ohio. Tanks so lined and sheathed were intended originally for steel pickling. They will do as well for leaching lead, zinc, and copper ores, for evaporating salt brines, for making various chemicals, and for other purposes. Most of these applications were formerly considered too severe for rubber lining. The combination of lining and sheathing is ideal, according to the manufacturer. The resistance of Triflex to impact is said to be nearly 100 times that of ordinary rubber linings.

Testing Sieves for the Laboratory

A testing sieve shaker known as "End-Shak" has been developed by Newark Wire Cloth Company, Newark, N. J., to provide a shaker that would put through any given testing screen in a definite and shorter length of time all the material that would eventually pass. According to the maker, comparative tests have shown that the original purpose has been fulfilled. The new shaker weighs 275 lb., including motor and auto time switch, and stands 29 in. high. It will hold any number of 8-in. diameter sieves from one to thirteen inclusive.

An Oilproof Gasket Compound

A new substance, known as No. 1281 gasket compound and said to be proof against hot oil and gasoline, has been developed by General Electric Research Laboratory. It is a gray or brown, odorless and sulphur-free alkyd resin material, now called Glyptal. It is flexible, practically incompressible, and in outdoor exposure tests shows no noticeable hardening or stiffening. It becomes somewhat brittle at temperatures below zero Fahrenheit, but this does not affect its efficiency in service.

Industrial Notes

Schlumberger Electrical Prospecting Methods has moved from 60 Broad St. to 60 Wall Tower Building, 70 Pine St., New York.

Industrial Plants Corporation, 25 Church St., New York, is distributing a catalog listing of equipment being offered in liquidating the foundry, machine shop, and pattern shop of the Cresson-Morris Company, Philadelphia, manufacturers of machinery and transmissions.

Western Industrial Diamond Co. has opened offices in the Russ Bldg., San Francisco. The company imports carbons, bortz, and ballas.

Bulletins

Chain Drives. Renold and Coventry Chain Co. Ltd., Manchester, England. Catalog Ref. No. 016/34. Renold stock chain drives. In English, French, German and Spanish. Pp. 168.

Precision Measuring Instruments. Telefon-A. B. L. M. Ericsson, Stockholm, Sweden. Catalog U43. Measuring instruments for line measurements and laboratory use. Pp. 81.

Stokers. Patterson Foundry & Machine Co., East Liverpool, Ohio. Form P-178. Patterson stokers. Pp. 4.

Transmission Belting. B. F. Goodrich Rubber Co., Akron, Ohio. Supplement to mechanical goods catalog. "Highflex" transmission belt. Pp. 8.

Safety Equipment. Mine Safety Appliances Co., Pittsburgh, Pa. Folder. Protective hats and caps. Pp. 4.

Crushing Equipment. Traylor Engineering & Manufacturing Co., Allentown, Pa. Circular No. 87, describing new plant of Hudson River Stone Corporation. Pp. 8.

Measuring Instruments. General Electric Co., Schenectady, N. Y. Catalog GEA-1480B. Photo-electric recorder for recording any measurable quantity. Pp. 4.

Pumps. Northern Pump Co., Minneapolis, Minn. Bulletin 5000. Northern hydraulic pumps and motors of rotary radial piston type, in capacities 1 to 150 g.p.m.; 4,000-lb, pressure.

Grinding. B. F. Goodrich Rubber Co., Akron, Ohio. "Linerite" rubber lining for rotary grinding mills. Pp. 18.

Haulage. Cleveland Tractor Co., Folder No. 660. Model 80 tractor. Pp. 6.

Flotation of Gold Ores. Denver Equipment Co., Denver, Colo. Bulletin 3216 entitled "Flotation Opens New Opportunities for All Types of Gold Placer Mining." Pp. 4.

Screens. Link-Belt Co., Chicago, Ill. Catalog 1362. Vibrating screens.

Conveyors. Mining Engineering Co., Ltd., Worcester, England. Catalog. Specialties for underground conditions. Pp. 20.

Welding. Westinghouse Electric & Manufacturing Co., East Pittsburgh, Pa. Two leaflets describing new 300- and 400-amp. gasoline-engine driven Flex Arc welders. Pp. 2 each.

Modernization. General Electric Co., Schenectady, N. Y. Pamphlet, entitled "Modernization Pays." Pp. 12.

Gold Research. Denver Equipment Co., Denver, Colo. Bulletin 3201. Advances made in recent years in metallurgy of gold. Pp. 6.

Pulverizing, Air Drying. Raymond Bros. Impact Pulverizer Co., Chicago. Bulletin. Review of pulverizer installations. Pp. 16.

Electric Cable. General Cable Corporation, New York. A folder on the company's products.

Rubber. Diamond Rubber Co., Akron, Ohio. Bulletin. Mechanical rubber goods. Pp. 24.

TRENDS AND DEVELOPMENTS IN THE INDUSTRY

Broad Technical Scope Characterizes Institute Meeting

New York, N. Y.—At the annual meeting of the American Institute of Mining and Metallurgical Engineers, held in New York on Feb. 20-23, Dr. F. M. Becket, vice-president of the Electro Metallurgical Company, assumed office as president of the Institute, succeeding Dr. Scott Turner. Henry Krumb became first vice-president and Edgar Rickard second vice-president. Frank L. Sizer, of San Francisco, and Hugh Park, of Cobalt, Ont., newly elected to the board, took their respective places. Cadwallader Evans, John M. Lovejoy, Milnor Roberts, and J. V. W. Reynders were again elected as directors. Attendance at the meeting was good, though somewhat smaller than that of last year. The technical sessions provided the usual variety of papers. Some of these having particular interest are summarized briefly in the following.

Subsidence and Mining Methods

At the session of the Mining Methods Committee on subsidence, R. D. Parks, of the Michigan College of Mining and Technology, told how the yieldable metal prop developed by him for underground support a few years ago had recently been improved. It consists of two intersliding elements the telescopic action of which is resisted by a third element. The intersliding elements are rigid; the resisting element is yieldable. Metal and metallic alloys are used throughout. The yield point of the third element is below the column strength of the interslidable element. For laboratory purposes the prop consists of two pipes, of such diameters that one will slide within the other. The larger pipe, cast about one-half full of lead, serves as the base of the prop. The smaller pipe, with a regulatory plug fitted to the lower end, stands upon the lead inside this base. Under compressive loadings, the lead flows up through the upper pipe. Results are encouraging. Unlike those secured from previous tests, the curve plotted in a stress-deformation diagram shows that the load mounts rapidly to the yield point, drops about 10 per cent when the flowage of lead starts, and then continues throughout the full yield limit of the column at a uniform load. The maximum deviation of the curve was 2 per cent.

Any good machinist can make the prop and a good timberman can install it, according to Mr. Parks. Original cost compared with that of timber at present prices is relatively high, and is estimated to range from $4 to $14, depending on size. Replacement cost, however, is low, requiring only the recasting of the lead. Nothing is destroyed or lost when the lead yields. Barring accidents, the intersliding elements can be used indefinitely. The respective sizes of the top and bottom pipes and the plug can be standardized locally. The prop is fireproof.

Effect of approximately vertical cracks on the behavior of horizontally lying roof strata was discussed by P. B. Bucky, of the Columbia School of Mines, whose conclusions were based on experiments with small-scale models, such as he has used before, representing the roof and pillars of a mine. Such cracks in any number and position, where the sides of all cracks are in juxtaposition, decrease the roof strength by a maximum of less than 15 per cent, roof strength being here defined as directly proportional to the load it will support, he said. The presence of these cracks appears to increase the surface subsidence over that with a solid roof. A center crack causes less subsidence than one off center. Roof cracks of which the sides are not in juxtaposition materially weaken roof strength. Grouting or cementing of roof cracks whose sides are not in juxtaposition may strengthen the roof to a point equal to or greater than that of a solid roof, depending on the material used, its strength, and the bond strength between it and the roof material. An important factor governing roof strength is the degree of fixity of the roof at and beyond its supports. With solid roofs having a high degree of end fixity, surface cracks will tend to appear over the supports before they appear underground.

Subsidence at the Athens iron mine, in Michigan, was described by C. W. Allen, of the Cleveland-Cliffs company. Over part of the surface is quicksand, 150 ft. thick in places. Sublevel caving is used, but it was started at the deepest part of the (15 deg.) pitching orebody instead of at the shallowest. Surface subsidence has occurred, not over the deepest workings as planned but much nearer that point than it would otherwise have taken place, and the resultant delay in its appearance has permitted the mining plan to be carried out as originally decided on.

Iron mining received other attention also. The methods and costs of the Alan Wood Mining Company, in northern New Jersey, were discussed in two papers respectively by C. H. Loux and by H. M. Roche and R. E. Crockett. Mr. Loux dealt with the Washington mine, where a sublevel stoping system is used. Commenting, C. F. Jackson said that to those unfamiliar with substoping, the cost of stope preparation often appeared to be a serious objection. A study of costs at mines under a wide range of conditions showed that irrespective of hardness and strength of ore it varied but slightly and was only about 8 to 10c. per ton of ore developed. In a very narrow deposit it would be higher.

Extent of Bureau of Mines service to operators of small gold mines was recounted by C. W. Wright, of the Bureau. Several instructive bulletins, particularly I.C. 6611, now in its third printing, have been distributed. Washing gold gravels has not been very profitable to most of the untutored thousands who have tried it the past year. A press bulletin to this effect has been issued. Amount of gold recovered per individual has been estimated at 10c. to $2 per day. More important, in Mr. Wright's opinion, is the development of lode mines, as this will affect a larger group and require more plant equipment. The Bureau, he announced, has started a new series of papers on small-scale mining operations.

E. D. Gardner, of Arizona; A. M. Smith, of Nevada; and F. A. Thompson, of Montana, in subsequent discussion, thought it desirable to prevent further increase in the number of people who were flocking to the Western states to try mining.

Mining Geology

At the sessions on mining geology about a dozen papers were considered. George W. Bain, in a discussion of wall-rock mineralization along Ontario gold deposits, sought to establish a correlation between the grade of ore and the amount of the replacements in the wall. He cited instances from the Pearl Lake, Kirkland Lake, and Red Lake districts. Discussion showed that he and L. C. Graton were not of one mind as to the character of the replacements, the latter holding that they were invariably of chemical nature. Mr. Bain thought that some were caused mechanically.

D. H. McLaughlin and Reno H. Sales presented informally a chapter from the Lindgren Volume, the topic being the "Utilization of Geology by Mining Companies." C. S. Blair, of the Tennessee Coal, Iron & Railroad Company, offered a paper on the relation of the mining geologist to the mining industry in the Birmingham district. Sidney H. Ball said that a future program should give space to consideration of the relation of the U. S. Geological Survey to the mining industry.

O. H. Hershey discussed mining geology in the Coeur d'Alenes. Harrison Schmitt presented two papers on Southwest geology. T. M. Broderick, of Calumet & Hecla, spoke on applying geology to iron-ore concentration problems. Paul Billingsley and Augustus Locke also had place on the program with a joint paper on the tectonic position of Rocky Mountain ore deposits.

Walter Harvey Weed spoke a word on the rôle of volatiles in ore genesis. All magmas contain gaseous constituents, he said, which play a part in ore genesis. These have long been held to have but a minor part in the formation of ore deposits, save for those of tin and some others. But it is now recognized that the volatiles are not only transporting agents but also have a part in fractional crystallization and in all gaseous and explosive activity. Dr. Weed intends to publish a written paper on this theory.

Geophysics

Geophysics drew its quota of interested members. Among the papers were two by C. and M. Schlumberger and E. G. Leonardon, one of them being entitled "A New Contribution to Subsurface Studies by Means of Electrical Measurements in Drill Holes." C. A. Heiland and W. E. Pugh presented a lengthy paper on certain instrument and field problems in reflection seismology.

Results of a preliminary magnetic survey of a gold placer area in Montana were offered by Mr. Pugh. The locality was Lincoln Gulch. The gravels have a relatively low proportion of magnetite. A Schmidt vertical intensity field balance which measures changes in the vertical component of earth's magnetic field was used. About 400 stations were occupied, whereas 5,000 should have been to survey properly the 1,240 acres involved. Results were not interpreted quantitatively. A careful survey, it is expected, will delineate the position of the channels and, with the aid of a minimum number of check drill holes, will aid in exploiting the area at a cost of a fraction of that of detailed core drilling.

Health and Safety in Mines

Interesting figures on the prevalence of dust diseases in mines, including silicosis, were presented by M. van Siclen, of the Bureau of Mines. The seriousness of the dust-disease situation is being increasingly recognized by State and company officials. Agencies for study and prevention are many, but advance has been slow. Mr. van Siclen gave figures for the Tri-State district; Butte, Mont.; and Lead-Deadwood, in South Dakota. Butte is perhaps the most severely affected, considering the number involved and the siliceous character of the dust. The high-silica gold and silver mines of Nevada, Colorado, and California have caused their share of dust diseases, but figures are lacking. Silicosis has been reported also from the Coeur d'Alenes, Idaho; the Globe-Miami and Ray districts, Arizona; Tintic district, Utah; and the Keweenaw Peninsula, Michigan; but in these districts the degree of dust hazard has not been ascertained quantitatively. Some respiratory trouble has been noted in the underground mines of the Lake Superior and Birmingham districts. According to the figures (based on the 1929 census) 62,268 men are exposed to the hazard, in greater or less degree, in the principal copper, lead, zinc, gold (lode), and iron districts of the United States. In the non-metallic mining industries, exclusive of tunnel and foundation work, 23,665 are exposed; in non-ferrous smelting and refining, 13,166. The hazard of silicosis is said to be practically absent in making cement, although other respiratory complaints exist.

Dr. R. R. Sayers, of the Bureau, commenting, said that the problem presented demands attention. Some one, whether the State or the individual, must pay for the damage caused by dust diseases. Control of dust locally is the important thing. The amount of dust in mine air can readily be determined with the konimeter, which anyone of intelligence can quickly learn to use. Daniel Harrington pointed out that all dusts are harmful.

Non-ferrous Metallurgy

At the session on non-ferrous metallurgy, R. S. Dean, of the Bureau of Mines, summarized a paper by H. A. Doerner and C. G. Maier, recounting the progress made in reducing zinc with natural gas. After a false start on too large a scale, study has revealed the possibility of solving the major difficulties. Further demonstration of its possible application to continuous production is thought desirable by the principals, but is not now feasible, owing to lack of funds. The odds on the success of the process were said to be better than even. In a test run made last July, an extraction of 82.5 per cent was made. Recently, according to Mr. Dean, as high as 97 per cent had been attained. A long written discussion by C. A. Hanson, of Joplin, was read by the chairman.

J. O. Betterton described his process for dezincking lead continuously with liquid chlorine. The byproduct, zinc chloride, has an economic value, in contrast with that of the Harris process. Another advantage is the low installation cost, only a small amount of equipment being required. The equipment, moreover, is protected against corrosion by keeping the temperature in the reaction cylinder down to 750 deg. F., only 10 deg. higher than that of the lead in the bath. Temperature control is effected by pumping the lead fast enough (10 tons per minute) to dissipate the heat of the reaction between the zinc and the chlorine. The latter is absorbed at the rate of 500 lb. per hour. The zinc chloride, an anhydrous salt 99 per cent pure, is skimmed as fast as formed. The remaining 1 per cent is basic zinc chloride. Dezincking can be done in 8 hr. Chlorine being available at a constantly decreasing price, the process appears to have a future.

Milling Methods

The program on Milling Methods included a session on mineral physics, the papers except one being provided by the Bureau of Mines; also a session on flotation. Among the papers presented at the latter was an experimental study of certain reagents, by I. W. Wark and A. B. Cox; a paper entitled "Microscopic Examination of a Flotation Product," by A. M. Gaudin and H. B. Henderson; another entitled "Development of Improved Fine Grinding and Classification Flowsheets," by K. C. MacArthur and A. Anable; and a paper on the cyanidation of certain calcined gold ores, by Messrs. Leaver, Woolf, and Jackson. Two papers were contributed by A. W. Fahrenwald and associates. At a session on coal preparation, E. A. Holbrook discussed the relationship between ore dressing and coal preparation.

At a joint session of the Milling Methods and the Eastern Magnetite committees, G. I. Thatcher described a new, submerged-belt, wet magnetic iron-ore separator installed in the Scrub Oak mill, in New Jersey, and H. M. Roche and R. E. Crockett discussed the milling method used at the same plant. Two papers on phosphate recovery, by H. S. Martin and C. E. Heinrichs respectively, were presented. F. J. Tolonen described gravity concentration tests on Michigan iron formations with the aid of heavy fluids and suspensions as separating media. This work has not been completed, but the tentative conclusions show advantages to be gained by the use of these media. Mr. Tolonen finds the difference in specific gravity between chert and iron oxide minerals not great enough for the best gravity separation by the usual methods using water as medium. Arthur Hecht discussed the microscopic examination of an iron ore.

Non-metallics—Rare Minerals and Metals

The status and trend in the domestic potash industry were indicated in papers presented at the session on non-metallic minerals. Other topics treated were the use of sulphur, lime, and gypsum in agriculture; phosphate recovery; and Western fluorspar (by E. F. Burchard). Some notes on the economics of the non-metals were presented by Paul M. Tyler and Oliver Bowles. A paper by A. H. Emery dealt with mineral fillers for sheet-asphalt paving mixtures.

Sidney H. Ball presented at the session on rare minerals and metals a paper of much human interest on the mining of gems and ornamental stones by the American Indian. The metals calcium, strontium, and barium were discussed by Charles Hardy and Mr. Tyler.

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ALASKA AND NORTHWEST

Alaska Juneau Profits. Alaska Juneau reports 337,670 tons mined and trammed to mill in January, 1933, for a return of 73.74c. per ton. Operating expenditures totaled $187,500, or 55.53c. per ton, permitting an operating profit of $61,500, or 18.21c. per ton. Non-operating income, in addition, totaled $5,100. . . . Treatment charges for lead concentrate shipped to the smelter at Trail, B. C., will be reduced $1 to $1.50 a ton, by a reduction in the base rate and in the penalty for sulphur content. . . . Ophir Gold, near the Gnome Gold mine and between Elk City and Orogrande, Idaho, is being prepared for development and production this spring. Plans are being completed for the erection of a 60-ton mill, to be expanded later to 120 tons. . . . Briggs and Borcherding, operators of the Jingo mine in south central Idaho, 16 miles north of Hill City, have opened a promising oreshoot on the big Jingo vein for 200 ft. on the 50-ft. level. The Jingo is south of several parallel veins on the margin of the Idaho batholith. The veins run from 4 to 11 ft. wide, with high-grade oxidized ores. Sulphides extend virtually to the surface.

CALIFORNIA

Gold Activity Widespread. Operations are to start soon at the Erin Go Bragh mine, at Stent. The property has been equipped with electric-driven machinery required for large-scale development work. . . . Carnation Mining, operating the old Jamieson mine, at Johnsville, Plumas County, continues exploring ground west of the main tunnel. Production is to be resumed when weather permits. . . . Satisfactory results are being obtained at the Alpine mine, in the Georgetown district, where a new 200-ton mill was placed in operation recently. . . . Deepening the main shaft an additional 200 ft. and extensive lateral developments are to be undertaken by the Spring Hill Gold Mines, near Grass Valley. . . . Development work carried on at the Eureka-Woodside and Beebe mines and enlargement of the Beebe mill are proceeding satisfactorily, according to word received from Georgetown. Both properties are controlled by Atolia Syndicate, a Bradley subsidiary.

New Mills Planned and Being Erected. Montana-Nashville Mines, headed by J. A. Norden, general manager of Utah-Apex Mining, has acquired the Montezuma mine, near Plymouth, Amador County, and is moving in additional equipment in preparation for the reopening of the property. A 100-ton mill is planned. . . . Five additional stamps and a flotation unit are being installed at the mill of Wallberg Mining, near Challenge, Yuba County. Underground work recently started includes the sinking of a two-compartment shaft in the main tunnel to a depth of 300 ft. . . . Consolidated Gold and Metals, controlled by the Walker interests, of Salt Lake City, Utah, is engaged in developing its newly acquired property near Spring Garden. Installation of mining machinery and a power line has been completed recently. . . . Installation of the first 100-ton mill unit is scheduled this month by Madre de Oro Mining at the Porter mine, near Oroville. . . . Klamath Gold Placer is completing plans for the early operation of a leased mine, on Klamath River, near Happy Camp. The company controls 2,240 acres adjoining the old Siskiyou mine. F. H. Vahrenkamp and S. F. O'Leary, of San Francisco, will be in charge of operations.

MONTANA

New Strike at Basin Goldfields. In spite of winter weather and heavy snows, activity still continues in the Basin district in Jefferson County. At the Basin Goldfields property, of which a working control is held by Butte Copper Consolidated, a new orebody has been opened that assays over 7 oz. gold per ton. . . . On Cataract Creek, Morning Glory Mines has been reorganized. Carl Anderson has resigned as president and general manager, and the office of president is temporarily vacant. Reginald Rowand, Helena, has been elected general manager; David K. Rowand, Helena, vice-president; L. F. Hachez, Spokane, treasurer; and Brookes Sharpe, Butte, secretary and superintendent of operations. . . . Milling is to be started by Spokane Idaho Copper, on its gold property southeast of Libby, Mont. The plant was expected to be in condition for operation by Feb. 15. Capacity is 75 tons daily. . . . Montana Sunshine Mining, with ore developed on several levels, is preparing for the installation of a 200-ton concentrating mill. The mine is on Grouse Mountain, near Troy, Mont. The ore is of lead, silver, and zinc, with lead said to be predominating on the lowest levels and zinc on the upper levels.

UTAH

Chief Consolidated to Change Mill Flowsheet. According to a report by the general manager, Cecil Fitch, Chief Consolidated, with properties at Eureka, operations at which have been suspended since February, 1932, plans to remodel its mill, introducing changes suggested by G. H. Wigton, metallurgist, that are expected to add materially to future profits. The company is a producer of gold, silver, lead, copper, and zinc. The present plant, erected in 1924, treats the ore by differential flotation and table concentration. A volatilization process is used on the table tailing.

ARIZONA

Suspension Plans in Effect. All the larger mines of the Globe-Miami district are now inactive. Miami Copper was shut down for an indefinite period on Feb. 15. The skips have been unbent from the hoisting ropes and removed from the shaft, and the ropes have been coiled on the hoist drums. Employees have been discharged, except for a skeleton crew to guard the property, maintain the mine openings, operate the pumps, and to do such repair and miscellaneous work around the plant as may be necessary. The Old Dominion mine has been furnishing water to Miami Copper for milling purposes and to the city of Globe, but will not continue to run its pumps after losing the mining company business. Globe is therefore overhauling the old city pumping plant and will pump its own water. The Globe and Miami Reconstruction Finance Corporation organizations have on file upward of 3,000 applications for work from the unemployed of the district. These men are given work, chiefly on the highways, up to three days each, per month, at $4 per day. The amount of work each man gets is based on the number of his dependents. Miami Copper is allowing its employees to cultivate the tract of land in use for that purpose since War days, and is this year extending the privilege to ex-employees. Inspiration is allowing its employees, past and present, the use of the garden tract placed in use for its men last year, and in addition is preparing a 60-acre tract that will be divided into garden plots for anyone who cares to apply. Water for irrigation is furnished free by both companies.

Monte Cristo Property Changes Hands. The Monte Cristo mine, in Yavapai County, has been sold to Monte Cristo Gold-Silver, of Utah. The mine was sold in 1926 by Ezra Thayer to C. C. Julian for a reputed price of $1,000,000. Since then the property has been tied up in litigation most of the time. T. G. McKesson, of Phoenix, obtained title to the property through court action and sold it to the Protective Association of C. C. Julian investors, who, in turn disposed of it to the Utah company. The Monte Cristo is known as a silver mine, but is said to have gold-ore reserves. The Julian investors are receiving stock in the Utah company in payment for the mine. Russell S. Callister, treasurer of the new firm, states that his company bought the Monte Cristo in the expectation that the demand for silver will improve soon. As soon as the mine is ready for production a mill will be built. Workings on the property consist of an 1,100-ft. inclined shaft with levels at 100-ft. intervals and 22,000 ft. of drifts and crosscuts. An electric power line connects the mine with Prescott. . . . Belmont Copper Mining has granted five leases on its property near Superior to local miners, fifteen of whom are working on the leases at present. The values in the ore produced are chiefly in gold, with silver, lead, and copper. Another lease has been taken on the adjoining Queen Creek Copper Company ground, where siliceous gold ores are being mined along a contact between limestone and quartzite. The contact extends north into the ground of the old Lake Superior & Arizona Company, now owned by Magma Copper, which also has leasers working on the contact.

COPPER COUNTRY

Tribute to a Native Son. The death of John D. Ryan, chairman of the board of directors of Anaconda Copper, on Feb. 11, caused deep regret in the Michigan copper district, for the little locality of Pewabic, outside the city limits of Hancock, was his birthplace. Mr. Ryan's father was the late Capt. John Ryan, who was mine captain at the old Franklin mine and other mines in the Copper Country. John D. Ryan was employed for some time in the store of his uncle, the late Edward Ryan, in Hancock, and later engaged in the tailoring business. From there he went to Colorado, where he acted as representative for the Crew-Levick Oil Company, and later took a position as state representative for the Standard Oil Company in Montana. It was while in Montana that he became associated with the late Marcus Daly, then engaged in the development of Anaconda. Despite the fact that business matters kept him in the East, Mr. Ryan found time to spend a part of each summer in the Copper Country. He always enjoyed his visits and especially when he could reminisce with old-timers who knew or worked with his father. His last visit was during the holidays at Christmas, 1932. Reserved but genial, Mr. Ryan always maintained his wide acquaintance and had countless friends. He was always mindful of local institutions. St. Joseph's Hospital, in Hancock, was one of the many objects of his generosity. He assisted in financing the construction of the new nurses' home built only a few years ago, and also the hospital chapel.

New Crusher Proves Efficient. The new impact crusher installed by Copper Range in its Champion mill has worked effectively in preliminary trials. It was

built according to plans designed by Copper Range. Although it has not been operated at full capacity, the machine's demonstrated efficiency seems to indicate it will crush rock in half the time taken by the steam stamps, with less replacement and power expense. Mass copper is easily removed. Furthermore, the impact of the crushing shoes cleans the masses of adhering rock. The new crushing method is expected to effect an important reduction in operating costs. . . . Announcement that Mohawk Mining will submit to the annual meeting of stockholders on March 28 a proposal to dissolve the company was received with regret in the Michigan copper district. Mohawk, which stopped work last fall, was the lowest-cost producer of the Michigan mines and one of the most profitable. It was a substantial property, working steadily and employing at its peak about 700 men, underground, on surface, at the stamp mills and on the Mohawk-Gay railway, which the company owns. Thriving communities grew up at Mohawk and at Gay, which will be hard hit as a result of the cessation of operations. . . . The copper mining companies, as well as the producers of iron, coal, limestone, gravel, oil, and lumber in Michigan, will be subjected to a tax of two-tenths of 1 per cent on the gross value of the product at the time of sale if Governor Comstock's $35,000,000 general sales tax bill is enacted into law.

IRON COUNTRY

Vacuum Drying of Concentrate Effects Savings. At the Minnesota School of Mines experiment station at Minneapolis, iron-ore concentrate from a Dorr bowl classifier, averaging 25 per cent moisture, is being dried on a commercial scale with an Oliver vacuum filter, to determine saving in freight charges possible. In a one-unit plant producing 2,000 tons of iron concentrate in 10 hr. the bowl classifier product per day of two shifts totals about 1,200 tons, 60 to 200 mesh. The freight rate between Mesabi range and Lake Erie ports is $1.74 per ton, so with an estimated 10-per cent reduction in moisture by drying, the saving per ton is 17.4c. An average year's operation of 150 days at 1,200 tons per day totals 180,000 tons; by saving 17c. per ton by drying, the yearly profit is $30,500, which means a saving of over 100 per cent on the investment, as the total cost of the drying unit will not exceed $20,000.

NEVADA

Gold Find in Erstwhile Silver Mine. Discovery has been made at Rosebud, in northern Pershing County, of a vein of high-grade gold ore in the Brown Palace mine, once noted for ore rich in silver. The vein was found by John Etchart, one of six miners operating a lease in the old workings. It has been exposed for 40 ft., and at 15 ft. depth 8 ft. wide. A grab sample from 100 sacks of selected ore ran 7.28 oz. gold and 9.27 oz. silver. The leasers, known as the Raggold Six, have an option to purchase the property from Paul Webster, of Santa Monica, Calif. The Brown Palace was located early in 1907, by Phil Blume, of National, and was sold by Blume to Malcolm MacDonald, of Tonopah, for $100,000. MacDonald paid cash for the property and later abandoned it after working out a shoot of silver ore, which bottomed on a horizontal fault. Rosebud is 15 miles southwest of Jungo, on the Western Pacific Railroad, and 6 miles northwest of Scossa—a new gold camp that has attracted much attention. Charles Scossa, who found rich gold ore at Scossa late in 1931, was the first locator at Rosebud.

Placer and Flotation Plants Expand. Landis Mining, a Nevada corporation, is enlarging its plant for the recovery of gold at Rabbit Hole, in Pershing County, to handle 500 cu.yd. of placer gravel in eight hours. The mine is in Rosebud Canyon, and the auriferous gravel is hauled to the plant in trucks, washed and screened, and passed over a shaker table equipped with Hungarian riffles to recover the coarse gold. The table discharges to a 4x6-ft. revolving screen of 20 mesh, thence to a sliming table, which catches the black sand. Concentrated product from the tables goes to a Berdan pan amalgamator and thence to a mercury trough. The gravel averages $1 a yard; the black sand, $75, R. P. Landis, president of the company, reports. The deposit, 60 to 100 ft. wide, has an average thickness of 6 ft. It rests on a false bedrock of clay, and the power shovel that lifts the gravel is set to cut a few inches into the clay bed, to get all the gold. . . . Recovery Milling has completed installation of a ball mill and flotation plant at the McTigue amalgamating mill, at Silver City, on the Comstock. Capacity is enlarged to 50 tons. The milling company, J. L. Dobbins, manager, has a contract to mill ore from the Overman mine. . . . E. A. "Bob" Montgomery, veteran mine operator of Nevada, is developing an orebody 7 ft. wide in the old Keystone mine, on the Silver City branch of the Comstock lode. The first shipment, of 70 tons, to the Donovan mill, returned $14 a ton; the second, 100 tons, $18. Recovery was entirely by amalgamation. The ore is in a diagonal vein making off south and east from the Comstock lode. Values are almost entirely in gold. Operations are conducted through an incline shaft in the vein, which dips east at 55 to 60 deg. Montgomery has operated in Nevada for thirty years. In 1905 he located the noted Montgomery-Shoshone mine, at Rhyolite, which he sold to Charles M. Schwab. Two years later he developed the Skidoo mine, in the Panamint Mountains overlooking Death Valley, which became a large producer of gold. He has since operated mines in Mexico and New Mexico.

Court Decision Changes Coppermines Control. The Chancery Court of Delaware on Feb. 16 upheld the control of Consolidated Coppermines by the Higgins group, as against the Adee group. The court upheld the validity of the bylaws amendment increasing the number of directors from nine to fifteen, and dismissed all allegations as to fraud on the part of the Higgins group. It upheld the election of the eight directors elected by the Higgins group—namely, Ray W. Higgins, Thomas Bardon, Charles K. Blandin, William S. Gordon, for three years; Norman E. Lamond, Joseph B. Cotton, for two years; Samuel Brenner and Robert D. Hoffman, for one year. Messrs. Blandin, Gordon, Lamond, Cotton, Brenner, and Hoffman were the six additional directors elected under this amendment. The court upheld the election of Ray W. Higgins as president, Thomas Bardon as treasurer, and C. K. Blandin as vice-president, and found that the so-called rump meeting held by the Adee group which elected Howard D. Smith president, George T. Adee, vice-president, and E. J. McDonald, treasurer, was invalid. Therefore, it ruled that they have not been officials of the company since the last annual meeting in May, 1932, and it further restrained Messrs. Smith, Adee, and McDonald from interfering with the new officials in the discharge of their duties. A petition by the Adee group to obtain a stay in the court order giving the Higgins group full control of the company was dismissed. President R. W. Higgins has ordered an independent audit of the company's affairs, which is now being conducted. The Coppermines offices have been surrendered to the Higgins group. The decision reverses the findings of C. A. Sutherland, special master, who reported last November that the increase in the directorate was not legally made.

MEXICO

Iron-Ore Development Proposed. Victoria Mines, operating in the Altar district, Sonora, about 40 miles south of Nogales, Ariz., has increased its daily ore handling capacity to 100 tons as the result of leasing two 24-ton Huntington mills from Durango Mining. The former enterprise has its headquarters in Nogales, Ariz., and proposed to intensify its Altar district operations in March. . . . Elias Arias, former member of the federal senate from Colima state, is in Mexico City seeking to induce the national government to adopt measures for the systematic working of an iron deposit near Minatitlan, Colima. He estimates that the deposit can yield 12,000,000 metric tons, which after deducting operating and marketing expenses of 30 pesos the ton would afford a net profit of 10 pesos per ton, yielding the exploiters an annual profit of 1,840,000 pesos ($613,000, U. S.). . . . Negociacion Minera Iriarte will resume work in its La Cruz y Anexas mines, Pachuca, Hidalgo, late in March, after a suspension of more than two years following a dispute with its employees. The company made a pact with its former workers whereby they will be given employment preference and a 25 per cent share of the profits, in addition to their salaries. . . . Profitable occupation is being found in the Churunibabi mineral zone, about 5 miles north of Nacozari, Sonora, by more than 300 prospectors, many of them former employees of Moctezuma Copper. Desiderio C. Figeroa, a merchant of San Pedro de la Cueva, Sonora, is directing workers in a placer gold field near Rebeico.

State to Foster Mining Enterprise. To restore activities at Concepcion del Oro, Mazapil, and other mining regions along the borders of Coahuila and Zacatecas states, the Governor of Coahuila has invited Gen. Matias Ramos,

chief executive of Zacatecas state, and representatives of mining companies and their employees to a conference. Increasing facilities to mining companies by the state governments will be discussed. . . . The Sinaloa state government has appointed a committee of engineers to examine and report upon the gold fields surrounding the town of El Orito, Cosala municipality. The deposits, claimed to be rich, have attracted numerous prospectors from various parts of western Mexico. . . . A promising placer gold field has been uncovered by recent and prolonged torrential rains near the town of La Colorada, Sonora. The strike has attracted prospectors and jobless miners. Republican Mining & Metal is continuing to function at its antimony mines near Station Wadley, San Luis Potosi, at 25 per cent of normal capacity, and is working about 200 employees on a part-time service. Ore deposits opened up in the region are far in excess of present requirements, but the company is sending ore to a reduction plant in Laredo, Tex. Future plans of the company depend upon conditions that may develop in the American antimony market. . . . The mills of the A. S. & R. at Santa Barbara, Chih., are receiving daily between 60 and 80 metric tons of gold bearing ore from Southwestern Exploration's mines near Santa Barbara. The enterprise is working a vein ranging in thickness from 6 to 18 ft., and since August last has dispatched approximately 10,000 metric tons of ore to the mills.

CANADA

Gold Mining Successful and Active. The Kirkland Lake area continues to develop reserves of ore and productive capacity. Underground development is disclosing important deposits of ore that apex underground or are effectively concealed at surface by the waters of the lake and the deep overburden. . . . Toburn Gold Mines, formerly Tough-Oakes-Burnside, the pioneer producer of the district, resumed operations in 1932, with an output of $206,244 from 14,873 tons. On this, the most easterly producer on the "main break," the chances of future success seem good. . . . Sylvanite, next to the west, improved its ore position markedly in 1932 in the lower levels. Development is incomplete on the levels from 3,000 to 2,500 ft., but plans are under way to open four new levels down to 3,500 ft. Chances of finding a greater volume of ore below 3,000 ft. seem bright. Output in 1932 was $828,195 from 96,891 tons. . . . Wright-Hargreaves, with 4,000 ft. along the "main break," the largest holding, has made the most notable advance in recent months. In 1932 recovery was $12 a ton, and development of the levels from 2,500 to 3,000 ft. brought the reserves up to 951,939 tons, averaging $13.54. On the present lowest levels at 2,850 and 3,000 ft. the grade is over $20, and the orebodies are larger and more continuous than any so far opened in the mine. No. 4 shaft, being deepened to 4,000 ft., passed through rich ore recently at 3,900 ft. Preparations are now under way to increase mill capacity to 1,000 tons. . . . Lake Shore seems secure for some years to come as Canada's premier gold mine, as it is still a comparatively shallow property, with volume and grade of ore improving with depth. Output in 1932 was $12,647,128 from 818,698 tons. . . . Teck-Hughes continues to show the improvement in grade in its deepest levels recorded during recent months. Average recovery in 1932 was $12.40, with 475,700 tons milled, which is slightly above the average grade opened up so far in the deeper levels.

New Milling Plants. Kirkland Lake Gold, still the deepest gold mine in Ontario, has found more consistent bodies of ore at the 4,900-ft. level than on any level above, with average value of about $12. A winze is being sunk from the 4,900 level, to open new levels down to 5,450. In 1932, 56,592 tons was milled for a recovery of $524,329. Accumulated profits are now being used to develop the Kenty property, in the Swayze area, which gives promise of yielding the large volume of ore so far sought in vain on the older property. . . . Macassa, at the extreme west end of the "main break," now has sufficient ore of about $12 grade in sight on its lowest levels to warrant plans for a mill, probably of 200 tons' capacity. Contrary to the usual practice, this property has been developed from the bottom upward, access being gained conveniently from the adjoining Kirkland Lake Gold workings while a shaft was being sunk. The shoots of ore are being followed upward by raises, and levels above 2,325 ft. will be established consecutively. . . . Equipment and supplies for the 50-ton mill of Central Patricia Mines is now being assembled. Canadian Airways has a contract for delivering a part of this material, the air route being 90 miles. The larger part of the 400 tons of freight is being taken in by tractor over a winter road about 100 miles long. All will be delivered before the breakup. The mine is partly developed to 500 ft., with 45,000 tons of $13.50 ore indicated. The mill is expected to be in operation about September.

Beattie Adopts Straight Flotation. Beattie Gold Mines has signed a contract for 1,500 hp. with Northern Quebec Power, at the rate prevailing in the Rouyn camp—about $60. The mill is almost ready to turn over, and the mine will be ready for production when the mill is tuned up. The main haulage level is to be at 500 ft. instead of 375 ft., and this has necessitated changes at this level, already served by two shafts. Plans to produce bullion on the property have been abandoned, and the milling will be confined to a 10 to 1 concentration by flotation. As the arsenic content of the concentrate makes it unacceptable to local smelters, it has been arranged to sell it to American Smelting & Refining.

BRITISH COLUMBIA

Magnesite Discovery Reported. Discovery of a large deposit of crystalline rock magnesite near Cranbrook, in East Kootenay district, is announced by Dr. C. E. Cairns, of the Geological Survey. The deposit, estimated to be 30 ft. thick, occurs within a formation traced for more than 4 miles. Analyses compare favorably with those of the more important commercial deposits in other countries. . . . The Barkerville area is attracting widespread speculative interest. Several established mining companies, including Britannia and Newmont, are undertaking exploration in the area. Recently, new companies have been incorporated to acquire and develop properties there. Burns Mountain Gold Quartz has acquired ten claims immediately south of Cow Mountain, on which Cariboo Gold Quartz is operating. . . . Spectacularly rich ore was found recently in stoping above the eighth level of the Pioneer mine, in the Bridge River district. This enrichment does not represent a pocket in an otherwise unproductive section of a vein. Recurrence of enriched sections is an encouraging feature of the vein system of this mine. . . . At the Homestake, near Grand Forks, being developed by Hecla Mining, gold ore is reported to have been found during sinking operations. . . . The Motherlode mill, acquired by Reno Gold Mines to replace the mill destroyed a year ago by fire, is in operation. Recoveries of from 94 to 97 per cent are being made, the ore running from $22 to $23. About 75 tons is being milled daily.

Exploration and New Gold Production. Noble Five Mines, owning the Noble Five silver-lead mine, at Sandon, recently acquired the old Athabasca, Venus, and Juno gold properties on Toad Mountain, near Nelson. Athabasca was first operated in 1897. The adjoining and connected Venus and Juno were leased last year, when exploration was carried on and 123 tons of ore, at over $10 per ton, was shipped. Exploration at lower horizons is said to present promising possibilities. Deep approach can be made advantageously from the Athabasca workings. . . . Production has been resumed at the Reno, in the Sheep Creek area of the Nelson district; 1,664 tons of ore was put through the remodeled mill in January, during the process of "tuning up." Average recovery was 95.1 per cent, bullion to the value of $21,000 having been produced, exclusive of gold to the estimated value of $9,000 retained in the circuit. The mill, acquired last year with the purchase of the Motherlode-Nugget mines, was not designed originally to treat sulphide ores. To meet the new requirements, changes made in the flowsheet include provision for primary coarse crushing at the mine; replacement of the battery by a ball mill, with classifier; cyaniding of the slime and sand in Pachuca tanks; concentration of sulphides with Wilfley tables; and treatment of cyanide solution by the Merrill-Crowe process. A new 800-hp. power plant and a rope tramway have been constructed. . . . Kootenay Belle Gold Mines has been incorporated to operate the KootenayBelle mine, also in the Sheep Creek area. This property was reopened last year by a syndicate which in the course of development extracted and shipped 250 tons of gold-silver ore that yielded a net return of nearly $30 a ton. The results of development were also distinctly encouraging. . . . Interest continues on the ledge-gold possibilities of the Cariboo district, where staking on the snow, now several feet deep, has been done on an extensive scale. One company is said to have staked 30 square miles of territory. This staking in the

case of some of the large companies is not so unjustifiable as might first appear; it is based on geological knowledge obtained from official sources and from observations made by scouts who were in the field last summer. Another recent incorporation to operate in this area is Cariboo Yankee Belle Mining, which has acquired the Yank's Peak group on Yank's Peak between the towns of Barkerville and Keithley. It plans to drive a crosscut tunnel to explore at depth veins exposed on the surface by trenching and shallow tunneling.

Granby Resumes After Strike. Fear that the operations of the Granby company at Anyox might permanently cease, following the declaration of a strike at the mine on Feb. 1, was removed by the resumption of operations on Feb. 16. The strike was engineered by Communist agitators who succeeded in organizing at Anyox a branch of the Mine Workers Union of Canada, identified with the Workers' Unity League of Canada—the Canadian section of the Red International of Labor Unions. Although over half the workmen refrained from joining this organization, the radical element succeeded in dominating the situation and delivered an ultimatum to the company demanding recognition of the union and a wage increase. These demands being refused, the strike was called. The Granby company for two years has been producing copper at a cost considerably above the market value of the metal. In maintaining operations in these circumstances it has been actuated by the desire to support the local community dependent on the continuation of work. To do this it has seriously depleted its cash resources. It was inevitable that the directors in New York should decide that, in view of the strike, they were justified in closing down permanently, and notice to that effect was sent to local officials. The president and general manager, Charles Bocking, promised the men that if they expressed a willingness to return to work he would use his endeavors to induce the directors to reconsider this decision. The Red element was forced to leave the camp; the loyal workmen signified their desire that work should be resumed, and Mr. Bocking's representations were successful.

EUROPE

Yugoslavian Operations Successful. Trepca Mines in the last quarter of 1932 made new records in almost every department of its activities, reducing costs, improving recoveries and the grade of concentrate, and increasing net profit by over 60 per cent. Operating results compare as follows (all tons are metric, and values and earnings are given in sterling):

	Quarter Ended Dec, 31	Quarter Ended Sept., 31
Tons mined	145,726	134,791
Tons delivered to mill	137,904	129,065
Tons milled	136,704	129,265
Average grade—Lead%	9.01	8.86
Zinc %	9.05	9.31
Silver (oz.)	3.15	3.09
Recovery—Lead %	96.08	96.07
Zinc %	90.97	90.90
Lead concentrate recovered	15,254	14,671
Assaying, Lead %	77.61	74.98
Silver (oz.)	25.47	24.48
Zinc concentrate recovered	22,270	21,877
Assaying, zinc %	50.51	50.03
Cost per ton ore (inc. London expenses)	14s. 4d.	16s. 9d.
Revenue per ton ore	£1 3s. 5d.	£1 2s. 8d.
Quarter's estimated profit	£61,866	£38,523

Ability of this new producer to earn at the rate of over 22 per cent per annum on its issued capital, as it did in the December quarter, when profit was over 38 per cent of gross income, should stimulate mining activities in the area in Yugoslavia where Trepca is operating. Trepca has acquired the Belasica Mines properties, comprising about 358 square miles in two areas that adjoin Trepca. Payment will be 59,667 full-paid shares of Trepca stock and an agreement to pay liabilities not exceeding £3,000, and also the cost of winding up the Belasica company. The property has several known zones of mineralization.

AFRICA

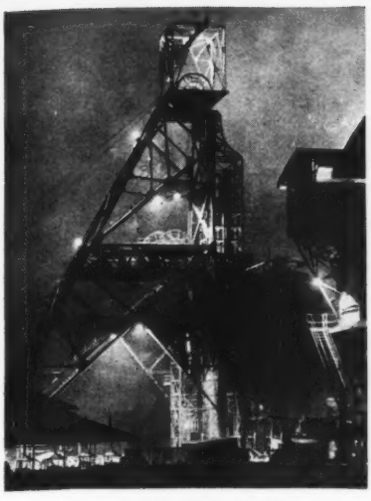
Future Rand Mining Policy. The Gold Producers' Committee, Transvaal Chamber of Mines, reporting on the Union's departure from the gold standard, states that a new mining policy will be evolved, under which advantage will be taken of the opportunity to lower the grade and increase the amount of ore reserves, accelerate development of orebodies now being worked, and undertake exploration and development of additional reefs, bringing into profitable production large quantities of ore previously regarded as unpayable. Inability to raise sufficient capital to sink additional deep-level shafts and for other major purposes has been a serious factor in limiting the scale of operations, restricting the exploitation of less promising reefs and areas and curtailing the prospective life of mines. Many programs can now be consummated by earmarking a proportion of the increased profits that should be earned in the future. Under the new conditions, although the actual increased price of gold on which the mines can rely remains uncertain, the estimated extra revenue to be derived by the Union Treasury from income tax and share of profits of leased mines, calculated on the current scale, will be about £5,000,000 per annum. This leaves out of account the enormous indirect contributions the industry makes to state revenues. The increased price of gold should greatly stimulate interest in the gold-mining possibilities of the Klerksdorp, Heidelberg, and other areas. The higher prices in local currency now obtainable for their products must also materially improve the outlook for diamond and base-metal enterprises, which will now be able to consider a resumption or expansion of production to a degree that will improve the economic situation and relieve unemployment.

Diamond Industry Under Strict Control. Sir Ernest Oppenheimer recently stated that, through the cooperation of the outside and conference producers, the Diamond Corporation was in a position to offer goods for sale only as the market required them, and steadily to maintain prices. The Government is working with the producers and the Diamond Corporation with this object. The diamond market is in a healthy condition. With the exception of the State diggings in Namaqualand, and the alluvial fields, all production of diamonds in the Union of South Africa and in South West Africa has ceased. The producers outside the Union have cooperated with producers in the Union in bringing about material reductions in output and prospecting work, and in reducing or ceasing deliveries of diamonds, and to this is due the improvement in the diamond market evident for the last three months.

Transvaal Chrome-Ore Prospects. Transvaal chrome ore was never expected to command the outlets that existed for Rhodesian ore, which is mainly high grade, assaying in excess of 48 per cent Cr_2O_3, whereas the Transvaal variety is low grade, assaying between 40 to 45 per cent, with a high iron content, which makes it useless for the manufacturer of ferrochrome. The Transvaal ore is limited to the manufacture of bichromate and furnace linings. Annual export reached its peak in 1929. Since that time a decrease has occurred in the quantity exported, partly due to the depresssion and partly to the Union's maintenance of the gold standard.

Outlook at New Kleinfontein. New Kleinfontein has for a considerable time been carrying on operations at a loss, during which period the Government has been remitting portions of its payments to the state, which has enabled the mine to carry on. Recently, several important developments have been reported, including a rich strike on the Brakpan-New Kleinfontein boundary, of over 500 ft. of development averaging 537 in.-dwts., the continuation of which area Brakpan Mines has offered to open up where it enters the Apex section of the New Kleinfontein. The company intends to increase development of this sction to 24,000 ft. per annum on the Apex section alone, which

Plant of Trepca Mines at Zvecan, Yugoslavia, producing lead and zinc and operating at a substantial profit



Roan Antelope's Beatty shaft

is higher than the total mine footage for nearly ten years.

Electrolytic Plant Resumes. Rhodesia Broken Hill Development has resumed operations of its electrolytic zinc plant in Northern Rhodesia, a reduction of freight rates having been secured and arrangements for the sale of the product completed. . . . Interest in the new Kakamega gold field in Kenya, formerly East Africa, continues to increase. Reports have been published of a struggle for control of the district, and also of trouble between whites and natives; but these appear to have little foundation in fact. The gold possibilities of Kenya are only beginning to be investigated. Good specimens have been obtained over widely separated areas, and several mining groups are interested. The Tanganyika Concessions group has an option on a large area, and has applied for an exclusive prospecting concession over an area of 6,000 square miles. Opinion seems unanimous that the field would be developed rapidly and thoroughly under the concession method.

Roan to Report Quarterly. Roan Antelope has discontinued the publication of monthly production figures, probably because Union Minière and Rhokana Corporation do not announce monthly production. Roan Antelope's operating costs for the last quarter of 1932, computed on the basis of $3.32 to £1, were as follows: Mining, 56.1c. per short ton; milling, 23.8c., and smelting 0.64c. per pound of blister copper. Including freight, London, and all other expenses except interest and depreciation, the total cost per long ton of copper was £22-8-2. The cost per pound of Roan Antelope's copper was, by an interesting coincidence, just 1/100th of the dollar value of sterling. Exchange averaged $3.32 and the cost of copper for the quarter was 3.321. That is, each 100 lb. of copper cost £1. With exchange at $3.43 as at present, the cost per pound would have worked out 3.431c. However this ratio is unlikely to continue, because the company is proceeding with the work of reducing its production cost, as indicated by the advantage gained by single-stage grinding, and it will be made slightly lower still this quarter because the Union of South Africa, where part of Roan Antelope's supplies are purchased, is off the gold standard. All Roan Antelope copper is finding immediate sale.

Sterling Profit Exceeds Par Yield. Ashanti Goldfields recovered 14,730 oz. gold in January, of an estimated value at par of £62,572. Working costs and general expenses in West Africa and London totaled £19,046. Premium on sales of gold amounted to £27,799, and estimated total profit for the month was £63,781 sterling.

INDIA

Silver-Lead-Zinc Output. Burma Corporation reports that 39,721 tons of ore was mined in January, 1933, for a production of 5,880 tons lead, 520,889 oz. silver, 5,090 tons concentrate assaying 8.4 oz. silver, 5.55 per cent lead and 53.4 per cent zinc, 1,300 tons copper matte, and 80 tons refined antimonial lead.

AUSTRALASIA

Gold Output Steadily Increases. Australian gold yield for 1932 was 700,761 oz., compared with 595,213 oz. for 1931. During 1933, Western Australia, with increased outputs from Lake View & Star and Wiluna, is expected to show a large increase, and Queensland, with Mount Morgan and Mount Coolon in full operation and returns from Cracow, should have an output much greater than that of Victoria in 1932. In Victoria much depends on developments in the Bendigo field. Many mines are being opened up on old Victorian fields, and these will make a substantial contribution to the 1933 yield. New gold mining companies formed during recent weeks include Hillbricks New Concord, to reopen the Concord line of saddle reefs between Newstead & Maldon, Vic.; Ararat Gold Mines Treatment, to acquire several deep lead leases at Ararat, Vic.; New Year Gold Mining, to develop a mine at Walhalla, Vic.; and Lisle Alluvial Gold, to prospect at Lisle, Tasmania, in behalf of a London company.

Wiluna Expands Operations. Wiluna Gold, Western Australia, has re-sampled its dump of cyanide residue, which can now be re-treated profitably. A small plant for this purpose will be in operation by the end of April. Underground ore, slightly lower in grade than originally estimated, is being met in development. The mine is now operating profitably. Estimates of capital expenditure required to increase the milling plant to 50,000 tons monthly have been prepared. The plant is now being extended to treat 40,000 tons monthly, and these additions will be complete by June, 1933, when recovery is expected to exceed 84 per cent. The arsenic section of the plant, which is now producing 3,000 tons annually, is to be improved by the installation of a Cottrell precipitator. . . . Goldfields Australian Development, recently formed in London, proposes to confine its attention for the present to the proving of large but low-grade properties near Wiluna, and at Laverton, Western Australia. . . . Sons of Gwalia has completed the repayment of a loan of £76,132 from the Western Australian government for the purchase of new power and milling equipment. The main shaft has been sunk to No. 26 Level, and driving is being carried out in ore of average grade. . . . On the Great Boulder property at Kalgoorlie, a diamond-drill hole from the 23-ft. level of Robertson's shaft has met three rich veins—No. 1, 72 in. wide, assay 8 dwt.; No. 2, 24 in. wide, assay 35 dwt.; No. 3, 24 in. wide, assay 42 dwt.

Mount Isa Seeks Freight Reductions. J. P. Webster, director of Mount Isa Mines, recently announced that, despite the premium on exported Australian lead, operations are still being carried on at heavy loss. The company has decided to obtain further capital and will not close unless the price of lead falls appreciably. Mount Isa is now milling 63,000 tons monthly of ore assaying 11 per cent lead and 5.4 oz. silver, and is producing therefrom 5,000 tons of lead bullion. Of the total cost of production, 40 per cent was accounted for by direct transport costs, of which more than one-half was for transport on the Queensland railways. The rates paid by the company, Mr. Webster said, were more than double those of similar companies overseas. As a result of negotiations, a substantial reduction in freight rates is expected. . . . At the end of April, 1933, Mount Morgan, Queensland, expects to be treating 3,000 tons of auriferous copper ore weekly, for 70 tons of concentrate. Profit is estimated at £40,000 per annum. Cost of milling is not expected to exceed 6/3d. per ton, and the total of mining and milling and all overhead and office charges, 20/- per ton. Mount Lyell Mining & Railway, Tasmania, has adopted deep-hole drilling to prove the low-grade disseminated orebodies of the Crown Lyell and Lyell Comstock properties, and is finding this method of underground prospecting particularly useful. Output of copper in the last quarter of 1932 declined by 1,100 tons, as a result of treating low-grade ore from development. . . . Electrolytic Zinc's output was slightly reduced during the early part of 1932 in conformity with the International Zinc Cartel. During recent months, however, production has been on an increasing scale, and for the four weeks ended Dec. 14 the output was 4,209 tons of electrolytic zinc, which indicates full plant capacity. Plans have been prepared to extend the plant from three to four units, but construction work has not yet commenced.

New Guinea's Gold Prospects. An important find of alluvial platinum and gold is reported from the eastern division of Papua, the metals being traced for 5 miles. . . . The crushing and cyanide plant of New Guinea Goldfields, completed in August last, has not yet reached capacity. Alterations in the flowsheet, as a result of complications that developed in the leaching of the slime portion of the ore, have been necessary. Discovery of two new and potentially important dredging areas is reported. One is 8 miles long by 1,500 ft. wide; the other, 10 miles long by 3,000 ft. wide . . . Bulolo Gold Deposits proposes to sample a large area of alluvial wash along the Koranga Creek and Bulolo River. . . . Bulolo Gold Dredging's working costs are now reported to average 6 gold cents per cubic yard.

Daily and Average Monthly Prices of Metals

February, 1933

United States Market

Feb.	Electrolytic Copper: Domestic Refinery	Electrolytic Copper: Export Refinery	Straits Tin: New York	Lead: New York	Lead: St. Louis	Zinc: St. Louis
1	4.775	4.700	23.350	3.000	2.875	2.850
2	4.775	4.700	23.450	3.000	2.875	2.800
3	4.775	4.725	23.500	3.000	2.875	2.800
4	4.775	4.700	23.500	3.000	2.875	2.800
6	4.775	4.725	23.700	3.000	2.875	2.800
7	4.775	4.725	23.675	3.000	2.875	2.625
8	4.775	4.725	23.575	3.000	2.875	2.700
9	4.775	4.700	23.750	3.000	2.875	2.600
10	4.775	4.800	23.700	3.000	2.875	2.650
11	4.775	4.850	23.700	3.000	2.875	2.650
13	Holiday	4.825	Holiday			
14	4.775	4.725	23.600	3.000	2.875	2.625
15	4.775	4.725	23.500	3.000	2.875	2.600
16	4.775	4.675	23.500	3.000	2.875	2.600@2.650
17	4.775	4.700	23.550	3.000	2.875	2.550@2.600
18	4.775	4.650	23.550	3.000	2.875	2.550@2.600
20	4.775	4.700	23.600	3.000	2.875	2.575@2.600
21	4.775	4.725	23.450	3.000	2.875	2.600
22	Holiday	4.675	Holiday			
23	4.775	4.700	23.400	3.000	2.875	2.600@2.650
24	4.775	4.675	23.300	3.000	2.875	2.625
25	4.775	4.675	23.250	3.000	2.875	2.625@2.650
27	4.775	4.650	23.200	3.000	2.875	2.650
28	4.775	4.600	23.200	3.000	2.875	2.650
Av. for month	4.775	4.710	23.500	3.000	2.875	2.666
Averages for Week						
1	4.775	4.717	23.167	3.000	2.875	2.900
8	4.775	4.717	23.567	3.000	2.875	2.754
15	4.775	4.771	23.650	3.000	2.875	2.625
22	4.775	4.688	23.530	3.000	2.875	2.593
Calendar Week Averages						
4	4.775	4.704	23.388	3.000	2.875	2.833
11	4.775	4.754	23 683	3.000	2.875	2.671
18	4.775	4.717	23.540	3.000	2.875	2.600
25	4.775	4.692	23.400	3.000	2.875	2.615

Silver, Gold, and Sterling Exchange New York and London

Feb.	Sterling Exchange: "Checks"	Sterling Exchange: "90-Day Demand"	Silver: New York	Silver: London	Gold: London
1	3.392500	3.402500	25.875	17.0000	121s 2½d
2	3.395000	3.406250	25.625	16.8750	121s 3½d
3	3.397500	3.406250	25.625	16.8750	121s 3 d
4	3.397500	3.405000	25.875	16.9375	.....
6	3.433750	3.441250	25.875	16.8125	119s 9 d
7	3.425000	3.433750	25.500	16.6250	120s 2 d
8	3.423750	3.433750	25.750	16.7500	120s 1½d
9	3.431250	3.443750	25.500	16.6250	120s 2 d
10	3.428750	3.441250	25.500	16.6250	120s 2 d
11	3.427500	3.438750	25.375	16.5625	.....
13	Holiday			16.5625	119s 10d
14	3.438750	3.452500	25.750	16.8125	119s 9½d
15	3.431250	3.450000	25.750	16.6875	119s 9½d
16	3.436250	3.456250	25.875	16.6875	120s 1½d
17	3.440000	3.450000	26.125	16.8125	120s 9½d
18	3.442500	3.460000	26.250	16.8750	.....
20	3.442500	3.458750	27.000	16.9375	120s 6½d
21	3.417500	3.432500	27.000	17.1875	120s 6½d
22	Holiday			17.0000	121s 8 d
23	3.416250	3.430000	27.250	17.3125	122s ...
24	3.407500	3.422500	26.625	17.3125	121s 10d
25	3.408750	3.425000	26.500	17.1875	.....
27	3.415000	3.435000	26.500	17.0625	121s 5 d
28	3.416250	3.446250	26.500	17.1250	121s 2 d
Av. for month	342.114	.....	26.074	16.885	.....
Averages for Week					
1	3.38979	.....	25.771	.....	.....
8	3.41208	.....	25.708	.....	.....
15	3.43150	.....	25.575	.....	.....
22	3.43575	.....	26.450	.....	.....

Calendar week averages: New York Silver, Feb. 4, 25.813; Feb. 11th, 25.583; Feb. 18th, 25.950; Feb. 25th, 26.875.

London Market

Feb.	Copper: Standard Spot	Copper: Standard 3 Mo.	Copper: Electrolytic (Bid)	Tin: Spot	Tin: 3 Mo.	Lead: Spot	Lead: 3 Mo.	Zinc: Spot	Zinc: 3 Mo.
1.....	28.3125	28.5625	32.750	148.5000	149.3750	10.6875	11.0625	13.9375	14.2500
2.....	28.5000	28.6875	32.750	150.0000	150.7500	10.5625	10.9375	13.5625	13.8125
3.....	28.3750	28.6250	32.750	149.6250	150.2500	10.6250	11.0000	13.5000	13.8125
6.....	28.3750	28.6250	32.750	149.7500	150.3750	10.5000	10.8125	13.6875	14.0000
7.....	28.4375	28.6875	32.625	149.6250	150.0000	10.3750	10.7500	13.7500	14.0625
8.....	28.5625	28.7500	32.750	148.8750	149.3750	10.3750	10.7500	13.7500	14.0625
9.....	29.0625	29.3125	33.000	149.6250	150.0000	10.4375	10.7500	14.0000	14.2500
10.....	29.1875	29.4375	33.250	149.7500	150.0000	10.4375	10.8125	14.2500	14.5000
13.....	29.0000	29.2500	33.250	148.3750	148.6250	10.2500	10.6250	13.8125	14.1250
14.....	28.5625	28.8125	32.750	149.0000	149.2500	10.2500	10.5625	13.8125	14.0000
15.....	28.2500	28.5000	32.500	148.5000	148.6250	10.1250	10.4375	13.7500	14.0000
16.....	28.2500	28.5000	32.250	148.6250	148.8750	10.2500	10.5625	13.8750	14.1250
17.....	28.3125	28.5000	32.000	148.2500	148.5000	10.3125	10.6250	13.9375	14.1250
20.....	28.7500	29.0000	32.500	148.2500	148.5000	10.5000	10.6875	14.0000	14.1875
21.....	28.3750	28.6250	32.125	148.2500	148.5000	10.5000	10.7500	13.9375	14.1250
22.....	28.3750	28.6250	32.500	148.2500	148.6250	10.5625	10.7500	14.0000	14.2500
23.....	28.3750	28.6250	32.375	147.7500	148.3750	10.5625	10.7500	14.0625	14.3125
24.....	28.4375	28.6875	32.250	147.3750	147.8750	10.5625	10.6250	13.9375	14.1875
27.....	28.2500	28.6250	32.250	146.2500	146.7500	10.3750	10.5625	13.7500	13.9375
28.....	27.8750	28.1875	31.750	146.2500	146.8750	10.3750	10.5625	14.0000	14.2500
Aver. for month	28.481	.....	32.556	148.544	.....	10.431	10.719	13.866	14.106

The United States quotations are our appraisal of the major markets for domestic consumption based on sales reported by producers and agencies. They are reduced to the basis of cash, New York or St. Louis, as noted. All prices of domestic class are in cents per pound.

Copper, lead and zinc quotations are based on sales for both prompt and future deliveries; tin quotations are for prompt delivery only.

Quotations for zinc are for ordinary Prime Western brands. Zinc in New York is now quoted at 0.35c. per pound above St. Louis, this being the freight differential. Contract prices for High-Grade zinc delivered in the East and Middle West are based on a minimum

Silver and Sterling Exchange

	New York		London Spot		Sterling Exchange	
	1932	1933	1932	1933	1932	1933
January	29.780	25.400	19.623	16.883	342.515	336.060
February	30.136	26.074	19.573	16.885	345.141	342.114
March	29.810	.....	18.336	.....	363.463	.....
April	28.298	.....	16.923	.....	374.731	.....
May	27.755	.....	16.868	.....	367.370	.....
June	27.466	.....	16.844	.....	364.471	.....
July	26.700	.....	16.930	.....	354.865	.....
August	27.986	.....	18.000	.....	347.449	.....
September	27.870	.....	17.998	.....	347.003	.....
October	27.195	.....	17.813	.....	339.500	.....
November	26.698	.....	18.099	.....	327.458	.....
December	25.010	.....	17.110	.....	327.683	.....
Year	27.892	.....	17.843	.....	350.137	.....

New York quotations, cents per ounce troy, 999 fine. London, pence per ounce, sterling silver, 925 fine. Sterling exchange in cents.

Copper

	F.O.B. Refinery			London Spot			
	Electrolytic			Standard		Electrolytic	
	Domestic		Export				
	1932	1933	1933	1932	1933	1932	1933
January	7.060	4.775	4.741	39.459	28.557	46.200	33.244
February	5.965	4.775	4.710	36.917	28.481	41.381	32.556
March	5.763	.....	.....	33.039	.....	36.786	.....
April	5.565	.....	.....	29.943	.....	34.190	.....
May	5.237	.....	.....	28.548	.....	32.833	.....
June	5.145	.....	.....	26.872	.....	30.841	.....
July	5.053	.....	.....	26.071	.....	29.107	.....
August	5.219	.....	.....	31.401	.....	34.784	.....
September	5.978	.....	.....	34.986	.....	38.318	.....
October	5.733	.....	.....	31.890	.....	36.190	.....
November	5.131	.....	.....	31.972	.....	36.568	.....
December	4.813	.....	.....	29.088	.....	34.344	.....
Year	5.555	.....	.....	31.682	.....	35.962	.....

New York quotations, cents per pound. London, pounds sterling per long ton.

Lead

	New York		St. Louis		London			
	1932	1933	1932	1933	1932 Spot	1932 3 Mos.	1933 Spot	1933 3 Mos.
January	3.750	3.000	3.550	2.875	15.084	15.128	10.458	10.833
February	3.712	3.000	3.499	2.875	14.560	14.571	10.431	10.719
March	3.150	.....	2.993	.....	12.345	12.634	.....	.....
April	3.000	.....	2.900	.....	11.223	11.503	.....	.....
May	3.000	.....	2.900	.....	10.673	11.036	.....	.....
June	2.993	.....	2.896	.....	9.608	9.898	.....	.....
July	2.747	.....	2.611	.....	9.818	10.152	.....	.....
August	3.235	.....	3.093	.....	11.349	11.588	.....	.....
September	3.465	.....	3.315	.....	13.122	13.349	.....	.....
October	3.052	.....	2.939	.....	11.958	12.170	.....	.....
November	3.050	.....	2.931	.....	12.071	12.395	.....	.....
December	3.000	.....	2.877	.....	11.144	11.541	.....	.....
Year	3.180	.....	3.042	.....	11.913	12.164	.....	.....

New York and St. Louis quotations, cents per pound. London, pounds sterling per long ton.

Tin

	New York		London	
	1932	1933	1932	1933
	Straits		Spot	
January	21.804	22.692	140.219	145.708
February	22.018	23.500	139.143	148.544
March	21.863	.....	129.810	.....
April	19.244	.....	108.935	.....
May	20.948	.....	122.286	.....
June	19.659	.....	114.530	.....
July	20.931	.....	125.863	.....
August	22.985	.....	142.017	.....
September	24.779	.....	152.705	.....
October	23.936	.....	151.280	.....
November	23.354	.....	153.585	.....
December	22.677	.....	149.806	.....
Year	22.017	.....	135.848	.....

New York quotations, cents per pound. London, pounds sterling per long ton.

Zinc

	St. Louis		London			
	1932	1933	1932 Spot	1932 3 Mos.	1933 Spot	1933 3 Mos
January	3.011	3.018	14.416	14.834	14.381	14.595
February	2.817	2.666	13.872	14.289	13.866	14.119
March	2.787	.....	12.616	13.024	.....	.....
April	2.725	.....	11.670	11.958	.....	.....
May	2.532	.....	12.432	12.682	.....	.....
June	2.777	.....	11.548	11.866	.....	.....
July	2.537	.....	11.592	11.967	.....	.....
August	2.758	.....	13.594	13.844	.....	.....
September	3.322	.....	15.455	15.616	.....	.....
October	3.027	.....	14.869	15.140	.....	.....
November	3.094	.....	15.264	15.534	.....	.....
December	3.124	.....	15.209	15.347	.....	.....
Year	2.876	.....	13.545	13.842	.....	.....

St. Louis quotations, cents per pound. London, pounds sterling per long ton.

Cadmium and Aluminum

	Cadmium		Aluminum	
	1932	1933	1932	1933
January	55.000	55.000	23.300	23.300
February	55.000	55.000	23.300	23.300
March	55.000	.....	23.300	.....
April	55.000	.....	23.300	.....
May	55.000	.....	23.300	.....
June	55.000	.....	23.300	.....
July	55.000	.....	23.300	.....
August	55.000	.....	23.300	.....
September	55.000	.....	23.300	.....
October	55.000	.....	23.300	.....
November	55.000	.....	23.300	.....
December	55.000	.....	23.300	.....
Year	55.000	.....	23.300	.....

Aluminum in cents per pound, 99 per cent grade.
Cadmium, cents per pound.

Antimony, Quicksilver, and Platinum

	Antimony (a) New York		Quicksilver (b) New York		Platinum (c) New York	
	1932	1933	1932	1933	1932	1933
January	5.976	5.722	64.900	48.500	40.000	26.480
February	6.489	5.738	66.304	48.614	40.000	24.000
March	6.188	.....	72.537	.....	40.000	.....
April	5.746	.....	72.125	.....	40.000	.....
May	5.170	.....	66.380	.....	39.500	.....
June	5.034	.....	59.481	.....	37.500	.....
July	5.000	.....	53.580	.....	35.200	.....
August	5.144	.....	47.444	.....	35.000	.....
September	5.606	.....	47.500	.....	33.800	.....
October	5.575	.....	47.600	.....	33.000	.....
November	5.771	.....	48.750	.....	33.000	.....
December	5.400	.....	48.500	.....	30.462	.....
Year	5.592	.....	57.925	.....	36.455	.....

(a) Antimony quotations in cents per pound, for ordinary brands. (b) Quicksilver in dollars per flask of 76 lb. (c) Platinum in dollars per ounce troy.

Pig Iron

	Bessemer		Basic		No. 2 Foundry	
	1932	1933	1932	1933	1932	1933
January	16.00	14.50	15.00	14.00	15.50	14.50
February	15.68	14.50	14.68	14.00	15.18	14.50
March	15.50	.....	14.50	.....	15.00	.....
April	15.50	.....	14.50	.....	15.00	.....
May	15.50	.....	14.50	.....	15.00	.....
June	14.85	.....	14.35	.....	14.75	.....
July	14.56	.....	14.00	.....	14.50	.....
August	14.50	.....	14.00	.....	14.50	.....
September	14.50	.....	14.00	.....	14.50	.....
October	14.50	.....	14.00	.....	14.50	.....
November	14.50	.....	14.00	.....	14.50	.....
December	14.50	.....	14.00	.....	14.50	.....
Year	15.01	.....	14.29	.....	14.79	.....

Iron in dollars per long ton. F.o.b. Mahoning and Shenango Valley furnaces freight to Pittsburgh, $1.89.

price basis of 4½c. per pound.

Quotations for lead reflect prices obtained for common lead, and do not include grades on which a premium is asked.

London prices for lead and zinc are the official prices for the morning session of the London Metal Exchange; prices for copper and tin are the official closing buyers' prices. All are in pounds sterling per long ton (2,240 lb.).

New York silver quotations are as reported by Handy & Harman and are in cents per troy ounce of silver, 999 fine. London silver quotations are in pence per troy ounce of bar silver, 925 fine. Sterling prices represent forenoon market demand. Cables command a premium.

Current Statistics of Production and Stocks of Copper, Lead, and Zinc

Data from American Bureau of Metal Statistics, American Zinc Institute, and Metallgesellschaft.

All Figures Except Tin Represent Tons of 2,000 Lb.

	1932											1933
	Feb.	March	April	May	June	July	Aug.	Sept.	Oct.	Nov.	Dec.	Jan.
COPPER—North and South America												
Production, refined, daily average	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Domestic shipments	.....	.....	.....	.....	.....	.....	(*a*) 18,300	(*a*) 19,700	(*a*) 25,500	(*a*) 25,000	.....	.....
Foreign shipments	.....	.....	.....	.....	.....	.....	(*a*) 48,000	(*a*) 53,700	(*a*) 55,500	(*a*) 55,000	.....	.....
Stocks, blister and refined	.....	.....	.....	.....	.....	.....	.....	(*b*) 775,000	(*b*) 773,000	(*b*) 770,000	.....	.....
LEAD — United States												
Production, refined, from domestic ore	28,081	30,345	23,236	25,902	26,068	15,819	17,118	20,498	21,092	24,465	21,173	24,615
Production, secondary and foreign	3,920	3,989	3,134	2,941	2,033	3,754	6,308	2,315	6,984	2,873	3,624	2,953
Production, total, daily rate	1,103	1,107	879	930	936	631	756	760	905	901	800	889
Shipments reported	26,319	31,162	26,081	24,258	21,511	19,723	28,248	26,413	29,764	22,838	24,089	19,030
Stocks, end of month	165,933	169,091	169,091	173,929	180,460	180,255	175,426	171,831	170,171	174,629	176,157	184,693
ZINC — United States												
Production, daily average	742	726	688	601	547	476	432	434	480	525	596	641
Domestic shipments	21,896	22,576	18,046	18,087	14,958	12,896	18,069	21,132	18,801	16,000	15,582	15,040
Export shipments	.....	.....	.....	.....	24	.....	39	20	20	20	20	20
Stocks, end of month	129,506	129,451	132,025	132,580	134,032	135,907	131,203	123,056	119,101	121,798	124,705	129,524
World Production Rate (Daily Average)												
Copper	.....	.....	.....	.....	.....	.....	.....	(*b*) 2,500	(*b*) 2,477	(*b*) 2,480	.....	(*b*) 2,260
Lead	3,872	3,598	3,587	3,711	3,497	3,190	3,334	3,323	3,510	3,460	3,350	.....
Zinc	2,615	2,623	2,528	2,392	2,343	2,224	2,121	2,120	2,255	2,343	2,381	2,540
Tin (Metric Tons)	274	340	312	311	261	178	179	255	249	258	239	.....

(*a*) Deliveries from all sources; private estimate. (*b*) *E.&M.J.* estimate.

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